MINE DISASTERS
AND
MINE RESCUE

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SECOND EDITION

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Universities Press

Universities Press (India) Private Limited

Registered Office
3-5-819 Hyderguda, Hyderabad 500 029 (A.P.), India
e-mail: *info@universitiespress.com*

Distributed by
Orient Longman Private Limited

Registered Office
3-6-752 Himayatnagar, Hyderabad 500 029 (A.P.), India

Other Offices
Bangalore / Bhopal / Bhubaneshwar / Chennai
Ernakulam / Guwahati / Hyderabad / Jaipur / Kolkata
Lucknow / Mumbai / New Delhi / Patna

© Universities Press (India) Private Limited 2007
First published 2007

Cover and book design
© Universities Press (India) Private Limited 2007

ISBN 13: 978-81 7371 573 0
ISBN 10: 81 7371 573 4

Set in Times New Roman 10/12 by
OSDATA, Hyderabad 500 029

Printed at
Orion Printers, Hyderabad 500 004

Published by
Universities Press (India) Private Limited
3-5-819, Hyderguda, Hyderabad 500 029

Preface to the First Edition

Disasters occur in mines as a result of large mine fires, powerful mine explosions violent rock gas outbursts, and water innundations. All operating mines have to contend with the real danger of being overtaken by one or more disasters at some points of time during their life. Plans for disaster management are, therefore, drawn by the mine managements and, therefore, the mining industry.

This book is a completely revised edition of an earlier work by the author *Mine Fires, Explosions, Rescue, Recovery and Innundations* which at once became a standard textbook on mine safety for all students of mining engineering in India. Owing to the rapid advances that have been made in mining research for further promotion of mine safety and the pressing demand for the book as it went out of market, the author undertook the arduous task of updating the book with new techniques, instruments, equipment, research findings, and experiences accumulated over the past decade. The subject of spontaneous combustion received an enlarged treatment. The inclusion of the subject of mine environmental monitoring and control which has become an important production tool and outbursts of coal and gas have added to the usefulness of the book.

Thanks are due to the following manufacturers of gas measuring instruments, environmental monitoring and control equipment, and mine rescue apparatus, who have generously supplied at different time points technical information with illustrations and schematic diagram for illustrating the text:

Drägerwork AG, Lübeck, Germany.
Auergesellschaft GmbH, Berlin, Germany.
Mine Safety Appliances Company, Pittsburg, USA
CSE Corporation, Monroeville, USA
J & S Sieger Ltd., Poole, UK
Hawker Siddley Dynamics Engineering Ltd., Welwyn Garden City, UK.
Oldham France SA, Arras, France.
Riken Keiki Instrument Co. Ltd., Tokyo, Japan.
Sabre Safety Ltd., Aldershot, UK.

Special thanks are also due to the Superintendent, Doncaster Mine Rescue Station, UK, and the Head of the Central Rescue Station, Essen, Germany who provided useful material for inclusion in the book.

Acknowledgement to the authors/information source is made in the appropriate places in the text.

Preface to the Second Edition

This edition of *Mine Disasters and Mine Rescue* has been brought out in the hope that it will continue to further the important cause of improving mine safety and mine environment in the Indian mining industry. While preparing this edition, steps have been taken to bring the material on mine fires, mine explosions and mine rescue, which is mostly of practical nature, up-to-date. The chapter on mine rescue has been re-written to accommodate the recent developments in mine rescue apparatus abroad and in India.

Although the book will be of great value to practising mining engineers and students of mining engineering, it is not really intended for specialists. Research has been continuously producing vast scientific and technical literature on the mechanisms of spontaneous heating of coal and sulphide ores as well as of other mine fires and explosions, all of which are easily available to the interested scholar. This book provides a basic though comprehensive treatment of the subject of mine disasters and mine rescue.

I wish to acknowledge the invaluable help provided by Mr. Nicholas Kyriazi of the US Bureau of Mines, Pittsburg Research Centre, who generously sent me product information and the latest reports of investigations on self-contained breathing apparatus and self-rescuers. I am also very grateful to Mr. William R. Flynn of Biomarine Inc, USA for making available to me useful drawings for inclusion in the book. I would like to acknowledge the generosity of Auergesellschaft mbH, Berlin; National Draeger Inc., Pittsburg; CSE Corp. Monroeville and Ocenco Inc., Kenosha, USA who provided me product catalogues on their mine rescue equipment. I am grateful for the encouragement given to me by Universities Press Pvt. Ltd. Last but not least I must thank my wife, Saroja, for her encouragement and enduring patience

M.A. Ramlu
2006

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The author wishes to express his gratitude to his son, M. Avanindra, an electronics engineer, who co-authored with him in writing the section on Environmental Monitoring & Control.

Appreciation is expressed to S.P. Hazra and Sri A.K.M. Burman of the Department of Mining Engineering IIT Kharagpur, for preparing the drawings with great care, ingenuity, and skill. Finally, the author express deep appreciation to his wife without whose driving force, the revision work would have taken much longer.

M.A. Ramlu
1991

To

my wife, Saroja,

for her encouragement and enduring patience

CHAPTER 1

Mine Fires

1.1 Underground Fires

Fires constitute an everpresent hazard in coal mines. They occur whenever and wherever combustible materials are present in mine workings. They endanger not only the valuable lives of men in a mine but also cause considerable economic losses to the organization affected by them. The losses incurred are both direct and indirect. The direct losses include loss of coal reserves and valuable mining equipment and the cost of firefighting and recovery of sealed-off areas. The indirect losses arise out of a fall in production from the affected section resulting in a fall in productivity and subsequent increase in operating costs. These costs are usually higher than the direct ones. It is well-nigh impossible to make an accurate assessment of the full cost of a serious mine fire. To those engaged in firefighting or in the unsealing of the fires, mine fires are far more hazardous than explosions.

Fires also occur in underground metal mines mining reactive sulphide ores.

Most mine fires start very small and can be extinguished, if detected in time, with a bucket of water or a bag of stone dust. It is only when a fire is discovered late or improper firefighting procedures are adopted that it gets out of control.

For a mine fire to break out, the following three conditions must be fulfilled:

(1) Combustible material must be available in sufficient quantity;
(2) Sufficient supply of oxygen must be available; and
(3) A source of ignition of adequate energy must be present.

If any of these conditions is not fulfilled, a mine fire will not break out.

There are two distinct types of mine fires:

(1) Incipient fires or fires due to spontaneous combustion; and
(2) Open fires.

Incipient fires, also called seam fires, are fires which originate in abandoned and discontinued mine workings, in goaves of seams being worked, in falls of roof coal, in fractured coal pillars, and in the vicinity of geological disturbances. They account for up to 75 to 90 per cent of all underground coal and 20 per cent of all underground metal mine fires, and are difficult to detect in their early stages.

Open fires are fires which originate in open mine workings such as shafts, roadways, and coal faces due to a variety of causes including spontaneous combustion and are characterized by visible active combustion or flame. They spread quickly in the direction of ventilation, the rate being dependent upon the kind and quantity of the combustible material, the amount of air reaching it, and the velocity of the air current. In any case, they develop much faster than incipient fires and spread quickly in the direction of ventilation if they are not suppressed within one or two hours after their outbreak.

Coal mine fires have a century-long history in Indian coalfields. The first fire was reported in the Raniganj Coalfield in 1865 and in the Jharia Coalfield in 1916.

1.1.1 Causes and prevention of coal mine fires

The various causes of coal mine fires may be grouped under the following main headings:

(i) Open flames
(ii) Spontaneous combustion
(iii) Electricity
(iv) Friction
(v) Blasting
(vi) Explosions
(vii) Miscellaneous

The relative importance of these causes varies from district to district in the same mine and depends to a great extent on the equipment and supplies used, on the training and discipline among the workmen, and on the liability of the coal to spontaneous combustion.

Despite various measures adopted to prevent fires in underground mines, fires still occur because of the above causes.

1.1.1.1 Open flames

In naked-lamp mines, in which naked or open lights are used, fires may originate through careless handling of lamps and carrying matches or other flame-making devices.

1.1.1.2 Spontaneous combustion

By the term 'spontaneous combustion', one understands self-heating of coal or of any easily oxdizable substance due to auto-oxidation at ordinary atmospheric temperatures.

Spontaneous combustion as an unwelcome feature had accompanied mining right from the days mining began up to the present day. There are several instances of fires due to spontaneous combustion in the history of mining which continued to burn for several decades and centuries. A well-known Indian example is the Jharia Fire consisting of several fires which has continued to burn since 1925. In reports of the shipping industry of the 19th century, cases of a number of coal cargo ships catching fire have been mentioned.

Today, storage of run-of-mine or prepared sizes of coal has become an integral part of coal-handling systems at mines, power plants, and metallurgical coking plants. The undesirable aspects of storage of coal, however, are spontaneous combustion and coal deterioration. Proper protection of coal storage piles is afforded by the prevention of oxidation of coal, which will prevent not only spontaneous combustion but also deterioration of coal.

The subject of spontaneous combustion has attracted the attention of several research workers since the beginning of the 19th century. It is still being worked upon today in many countries.

Most of the researchers have investigated the subject from chemical as well as physical considerations under isothermal and adiabatic conditions. The most important theories of spontaneous combustion that had been advanced are the Pyrite Theory (Plot), the Bacterium Theory (Potter), the Phenol Theory (Fischer), the Coal–Oxygen Complex Theory (Rhead and Wheeler), and the Electrochemical Theory (Kamneva and Aleksandrov). These theories do not hold good under all conditions and are applicable only under certain conditions because of the complex compositions of the coals and the lack of a full understanding of the mechanism of auto-oxidation. The generally accepted theory at present is the Coal–Oxygen Complex Theory which, according to a simplified consensus, considers that following an initial physical adsorption of oxygen, a variety of more or less stable coal–oxygen complexes are formed at different temperatures with release of gaseous products of oxidation.

The oxygen absorption reaction is considered to take place as follows:

$$\text{Coal} + \text{Oxygen} \longrightarrow \text{Coal–oxygen complex} \longrightarrow \text{Oxidized coal}$$
$$+ \text{CO}, \text{CO}_2, \text{H}_2\text{O}$$

The oxygen fixed on the coal is believed to be in carboxyl (–COOH), carbonyl (>CO) and other functional groups.

The common view, however, generally held by mining engineers is that the main causes of spontaneous heating are the sorption and oxidation properties of the coal substance itself at low temperatures. The following description is, therefore, suggested as a working hypothesis which fits the known facts.

Freshly exposed coal has, at ambient atmospheric temperatures, affinity for the oxygen of the air in contact with it. At first, the oxygen is adsorbed by a purely physical process, depending on the temperature, which, however, rapidly gives place to a chemical chain reaction resulting in the oxidation of certain constituents of coal with the production of a small quantity of heat.

A rise in the temperature of the coal, in consequence, takes place, which has the effect of further increasing the rate of absorption of oxygen and production of heat. If the heat produced cannot be dissipated by radiation or conduction or both at a rate faster than it is produced, a further rise in temperature takes place which accelerates further the rate of oxygen sorption and production of heat until, finally, the ignition point of coal is reached. It is evident from what has been stated above that the existence of a weak leakage air current is an important condition for spontaneous combustion to take place in mines. Such a condition exists in goaves of coal seams being worked by the bord and pillar method with open caving of the goaf or in the vicinity of faulted or geologically disturbed zones.

1.1.1.2.1 Factors affecting liability of coal to spontaneous combustion

Auto-oxidation of coal is a complex physico-chemical process in which concurrent and overlapping reactions take place which are difficult to separate out. The rate of oxidation of coal in air gives a measure of the liability of coal to spontaneous combustion.

The most important factors bearing on the liability of coal to spontaneous combustion are:

(1) *Surface area of coal* The rate of oxidation of coal and, hence, the amount of heat liberated depends on the total surface area of the coal including the internal surface area due to its pore structure. Pores are present in coal in abundance, so that the entire pore surface which, for different coals, is more than 80 m^2/g, forms the reaction site. According to Peters [1], free diffusion takes place in macropores below 10^{-5} mm in size, their velocities differing by a factor of 10^9. Münzner and Peters [2] found that, at low temperatures, the oxidation rates were practically the same for particles less than 0.3 mm in size and that above 0.5 mm, the deviations were significant. Kuchta et.al., [3] found from exploratory experiments that self-heating of most U.S. bituminous coals could be obtained with a maximum particle size of about 50 to 70 mesh (270 to 297 μ) at an air flow rate of about 50 cm^3/min.

(2) *Coal rank* The lower rank coals which are generally higher in moisture, oxygen, and volatile contents are more reactive and, hence, more susceptible to spontaneous heating than higher rank coals. The oxidation rate of coals of the same rank may, however, vary within a wide range.

(3) *Volatile content* It was found by several investigators that medium- to high-volatile coals (VM greater than about 18 per cent) oxidize faster than low-volatile coals and that high-volatile coals are more susceptible to spontaneous heating than are medium-volatile coals.

(4) *Petrographic composition of coal* Opinions were divided for a long time on the influence of petrographic composition of coal on the susceptibility to undergo spontaneous combustion. It is now generally accepted that the ease of oxidation of coal decreases with the macroconstituents in the order: vitrain, clarain, durain, and fusain.

(5) *Oxygen content* The increased reactivity of low-rank coals is attributed to their higher inherent oxygen content, and hence, the oxidation rate decreases with decreasing oxygen content, although the heating tendency is not in all cases proportional to the oxygen content.

(6) *Moisture content* The oxidation of coal in air at ambient temperatures takes place practically always in the presence of moisture in the surrounding oxidizing atmosphere, moisture inherent in the coal itself, or formed through oxidation. As the ability of coal to sorb moisture is considerably higher than that for oxygen or nitrogen, the H$_2$O formed due to the oxidation reaction is mostly absorbed in the coal, affecting its ability to absorb oxygen by hindering its diffusion and hence its oxidation rate. But a high inherent moisture and oxygen content, as found in low-rank coals and humidity in the surrounding atmosphere have been reported to be indicative of a tendency to spontaneous heating. Field experience has shown that the coal piles of Raniganj Coals (India) are more liable to spontaneous heating when the moisture content lies between 5 and 7 per cent. The exact mechanism of the action of moisture on the oxidation process in the early stages of oxidation, however, is not clear.

It is apparent that the presence of moisture would tend to reduce the tendency to spontaneous heating because of the latent heat of vaporization and high specific heat of water.

Studies conducted by the US Bureau of Mine on North Dakota lignite [4] have shown that the rates of oxidation were found to be maximum at 20 per cent moisture content and lower at either above or below this value.

(7) *Presence of pyrite* Iron pyrite (FeS$_2$) which frequently occurs in coal seams as discrete particles in a wide variety of shapes and sizes, distributed through the coal matrix, oxidizes easily at ordinary temperatures in moist air through a combination of reactions leading to the formation of ferrous sulphate and sulphuric acid besides other products, increasing in volume and liberating heat, in turn leading to accelerated oxidation and self-heating of coal.

The effect of pyrite on coal has been investigated by many workers with conflicting results. According to Münzner [5], the heat change caused by oxidation is the same for coal and pyrite in dry state and the reactivity of coal is only doubled when wet, whilst the reactivity of finely-dispersed pyrite (below 0.06 mm size) is raised 10-fold. The oxidation of pyrite in the presence of free moisture has only a promoting effect on auto-oxidation of coal when present in concentrations higher than 3%.

(8) *Ash content* Oxidation occurs chiefly on the organic portions of the coal and only slightly on mineral constituents. Ash in coal generally decreases the oxidation rate but may be influenced to some extent by the mineral composition of the ash, which is reported to have both accelerating and inhibiting effects on oxidation.

(9) *Temperature* Temperature exercises a great influence on the oxidation rate. Below ~ 70°C, the relationship between them is described by the Arrhenius equation. The rate of oxidation increases about two times for every 10°C rise in temperature. At a constant temperature, the oxidation rate will decrease with time and the oxidized products will accumulate on the coal. Lower temperatures favour accumulation of oxygen on the coal, whereas, at higher temperatures > 70°C, gaseous oxidation products are released.

(10) *Freshness of exposed coal surface* Freshly exposed surfaces of coal as when it is first mined consume oxygen at a relatively high rate which falls off continuously with time during the progress of oxidation for a given set of constant conditions.

(11) *Concentration of oxygen in contact* At any point on a coal surface, the oxidation rate is directly proportional to the oxygen concentration, that is, the partial pressure of oxygen surrounding the particles. For an atmosphere devoid of oxygen, the oxidation rate is zero regardless of all other factors. Coal may be stored in water indefinitely as without air there is no oxidation From laboratory studies conducted, the minimum oxygen concentration to develop spontaneous combustion has been been found to be 7 per cent [6].

(12) *Methane content of coal* The desorption of methane from coal creates an inert atmosphere in the macropores and micropores and may retard low-temperature oxidation, particularly in highly gassy coals. However, as the desorption of methane falls sharply with time, more and more of the coal surface becomes exposed to oxidation by air. The residual methane content of coal is reported to be the best indicator for predicting the hazard in a mine due to spontaneous combustion [7]. (Coals containing less than 5 m^3 of CH_4/t coal show high oxidation rates and are, hence, more liable to spontaneous combustion, while coals containing above 8 m^3 of CH_4 /t are not oxidized.)

(13) *Inhibitors* The rate of oxidation of coals and, hence, their tendency to spontaneous heating can be reduced by treatment with inhibitors or antipyrogenes which act in one or more of the following ways:

(1) Formation of a protective surface coating both on the macropore and micropore surface, resulting in the formation of a heat insultating layer;
(2) Formation of a layer which increases the thermal conductivity of the coal and thereby enhances the rate of removal of heat from local hot spots;
(3) Formation of a layer which consumes heat during its decomposition while the coal is in the process of oxidation before reaching the critical temperature range;
(4) Formation of a layer, preferably gaseous, which acts as a diluent and removes heat;
(5) Chain-breaking reaction which retards oxidation.

Many kinds of inhibitors – solid, liquid, and gaseous – have been tried by many investigators in the erstwhile USSR, Czechoslovakia, Germany, India, and a few other countries for fighting incipient and open fires in underground coal mines and coal-stack fires on the surface with a fair amount of success. Special mention may be made of $Ca(OH)_2$, $CaCO_3$, $Ca(HCO_3)_2$, NH_4Cl, Na_2SiO_3, $CaCl_2$, Montan Powder, NaH_2PO_4, $NH_4HB_4O_7$ and H_3BO_3

1.1.1.2.2 Contributing factors for spontaneous combustion of coal in mines

Besides the intrinsic properties of the coal, there are several other factors which contribute to spontaneous heating of coal in mines. These can be grouped under two main heads: geological and technical.

Geological factors

(1) Coal seams greater than 3 m thick pose the greatest hazard due to crushing of coal under rock pressure, high losses of coal during extraction, higher geothermic gradients in deep mines, and occurrence of bands of inferior coal.
(2) In zones of tectonic disturbances such as faults, the coal is in a crushed state, and leakage of water and air takes place along the fault planes when intersected by mine workings.
(3) Carbonaceous shales associated with the coal seams in the roof are a potential source of fires. The irregular bands of coal and unworkable seams occurring in the immediate roof and floor of a seam being worked may pose a danger, especially at great depths.
(4) Contiguous seams

The geological factors cannot be controlled.

Technical factors

Mining itself disturbs the coal seam and adjacent strata and can produce conditions that are favourable for self-heating.

The mining practices which contribute to the spontaneous combustion hazard are as follows:

(1) Extensive development within the seam (s) will expose a large area of the coal surface to oxidation as well as increase the leakage of air currents.
(2) The winning technology will determine the intensity of working of coal faces and the percentage recovery. A high rate of extraction is the best guarantee against the incidence of fires due to spontaneous combustion.
(3) The caving method of roof control and the orientation of fracture lines especially with bord and pillar or room and pillar workings control the stress concentration on the coal faces, pillars, and adjacent galleries and their crushing under pressure. Depending on the seam thickness, its depth, and nature of adjacent roof rocks, caving may develop cracks in the overlying seams unworked for whatsoever reason, in coal bands, and in left-over coals in the already extracted seams, creating conditions favourable for development of seam fires. Caving under shallow overburden cover may produce cracks reaching up to the surface, inducing leakage air currents from the surface to the underground.
(4) Unduly high ventilating pressure differentials between mine workings and across panels will cause air leakages through the coal and defective fire-stoppings, which might supply air just sufficient for oxidation but not sufficient for dissipating the heat produced. A good rule to be observed by mining men would be 'No ventilation at all is better than a deficient ventilation'.
(5) Mining of contiguous seams and thick seams in slices with danger of air leakage between goaves in the same seam and/or contiguous seams.
(6) Mining of seams liable to coal outbursts with the system of shockblasting, which creates conditions conducive to spontaneous heating through air leakage.

1.1.1.2.3 Stages of oxidation and spontaneous combustion of coal

The oxidation of different coals has been extensively investigated using thermogravimetric and thermovolumetric oxidation tests at low and high temperatures under isothermal conditions, but because of the diversity in coal substances and experimental conditions, one could draw only a few general conclusions out of the quite contradictory results which some of the studies have yielded.

According to the generally accepted simplified view of the complex low-temperature oxidation reactions, all coals react in the same basic manner, and that following an initial phase of physical adsorption of oxygen, a variety of solid coal–oxygen complexes are formed at different temperatures as a result of saturation at these temperatures of the unsaturated constituents of coal with the release of some gaseous products. A major proportion of the oxygen consumed is absorbed by most coals within the first hour. Physical adsorption of oxygen releases much less heat than does chemical oxidation of coal. Both CO and CO_2 are evolved at ambient temperatures immediately after the start of the oxidation, though with a slight delay interval between their liberation. Much more carbon dioxide is produced in the reaction than carbon monoxide. As the absorption of oxygen and temperature increase, CO increases slowly and gradually up to a certain temperature of about 40°C, beyond which it is evolved at an increasingly rapid rate.

At and beyond some elevated temperature between 50° and 100°C, the oxidation rate accelerates greatly due to a sustained exothermic reaction and ultimately produces smouldering or flaming combustion or ignition within the temperature range 327° to 420°C for bituminous coals. This temperature, usually 60° to 85°C, identifies the limit of low-temperature oxidation and the beginning of normal combustion under adiabatic conditions and is called the minimum or critical temperature or self-heating temperature. Hydrogen and hydrocarbons occur at temperatures above the critical temperature but their proportion in the oxidation products decreases with increasing oxidation.

1.1.1.2.4 Stages of spontaneous combustion of coal in mines

One recognizes three distinct stages of combustion of coal in coal mines:

(1) The incubations period;
(2) The indication period; and
(3) Open fire.

The incubation period is the period between the onset of first oxidation and the particular time at which one can detect it by the senses. From a practical point of view, the term, however, is used in a broader sense to denote the period between the first roof fall after beginning of coal extraction in a district or panel and the appearance of the first signs of heating. The incubation period varies widely, depending on the seam thickness, nature of the immediate roof, method of working, method of roof control, regularity and continuity of working, and liability of coal to undergo spontaneous heating. It usually varies between 3 and 18 months, higher rank coals having longer periods. During this period, one does not detect heating during one's passage through the mine workings.

The end of the incubation period marks the beginning of the indication period which is marked by 'sweating' and haze in the air, caused by the warmed-up air from a fire coming into contact with the cooler coal, rock, and metallic surfaces and depositing moisture. The indication period is often of very small duration lasting sometimes only a few hours. It comes to an end with the appearance of 'fire stink' when an open fire with visible active combustion breaks out. The fire stink can be easily recognized by the characteristic petrolic smell. Seams seldom burn with a bright flame but glow, developing bluish-white clouds of smoke.

1.1.1.2.5 Experimental methods for the study of spontaneous combustion of coal

Various methods have been used in the laboratory for studying the relative tendencies of coals to heat spontaneously. Most of the methods are based on the measurement of the oxidation rate and ignition temperature. The various methods may be broadly classified under the following heads:

(1) Isothermal method
(2) Adiabatic method
(3) Chemical method
(4) Temperature differential method
    (a) Crossing point method and
    (b) Differential thermal analysis (DTA).

Isothermal method (Isothermal calorimetry)

In this method, the coal sample is heated in a bath or furnace at constant temperature, with oxygen or air passing through it at a predetermined rate. The oxidation rate is determined, which can be used as an indicator of the coal's potential for spontaneous heating.

Adiabatic method (Adiabatic calorimetry)

The coal sample is placed in an insulated bath or oven chamber designed to maintain the coal and its immediate surrounding medium at the same temperature automatically so that no heat is lost or gained. During an experiment, the temperature of the sample is brought to a preselected value (starting temperature) in a stream of flowing nitrogen and after the system has attained thermal equilibrium, the apparatus is put into an automatic control mode, the nitrogen is shut off, and the sample is exposed to a flowing stream of preheated oxygen or air (dry or moist) when the temperature of the coal rises above its surroundings. The temperature of the bath/oven chamber automatically rises to coincide with the coal temperature. The experiment is run over a period of several hours up to 100 hours depending on the starting temperature and oxygen concentration till ignition occurs, which is determined as the minimum temperature at which there is a steep rise in the temperature on the temperature–time curve (Fig. 1.1). Coals

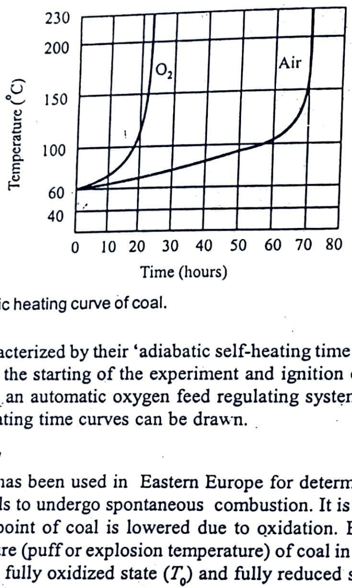


Fig. 1.1 Adiabatic heating curve of coal.

may also be characterized by their 'adiabatic self-heating time' which is the time interval between the starting of the experiment and ignition of the sample [8]. By incorporating an automatic oxygen feed regulating system, oxidation rate–adiabatic self-heating time curves can be drawn.

Chemical method

This method [9] has been used in Eastern Europe for determining the relative tendencies of coals to undergo spontaneous combustion. It is based on the fact that the ignition point of coal is lowered due to oxidation. By measuring the ignition temperature (puff or explosion temperature) of coal in the as-mined/as-received state *(T)*, fully oxidized state (T_0) and fully reduced state (the original *in-situ* state) (T_R), the degree of oxidation and liability index are calculated from the lowering of ignition temperatures as follows:

$$\text{Degree of oxidation} = \frac{T_R - T}{T_R - T_0} \times 100\%$$

Liability index $\Delta T_R = T_R - T_0$

When $\Delta T_R < 10$ no danger of self-heating
$10 < \Delta T_R < 25$ less liable to self-heating
$\Delta T_R > 25$ very liable to self-heating

Investigations conducted in the erstwhile USSR showed that the lowering of ignition temperature through oxidation is independent of the coal rank and that for coals of the same rank, it may vary significantly.

With this method, though the degree of oxidation of coal can be determined, there is some uncertainty regarding the link up of the puff temperatures with the ignition temperatures obtained by other methods.

The apparatus consists of a cylindrical copper block furnace with six holes for introducing quartz reaction tubes and a central hole for inserting a thermocouple which measures the temperature of the copper block and reaction tubes. This design ensures the same rate of heating for both the reaction tubes and the copper block. The open ends of the reaction tubes outside the furnace are connected through three-way cocks to six audiometer tubes placed in a trough filled with water. The audiometer tubes end in a common manifold at the top one end of which is provided with an aspirator bulb.

Two of the reaction tubes are filled with the original sample, another two are filled with fully reduced samples (treated with benzidene or p-phenylene-diamine) and the remaining two are filled with fully oxidized samples by treating, with perhydrol. Sodium nitrite is added to each of the samples and intimately mixed before filling into the reaction tubes. The furnace is switched on and the heating rate maintained at 4.5'C per minute. The water in the eudiometer tubes is raised to a certain level by means of the aspiration bulb, and the eudiometer tubes are then put in line with the reaction tubes. The temperatures at which a sudden fall in the water columns occurs due to rapid evolution of the reaction products are noted and the liability index calculated.

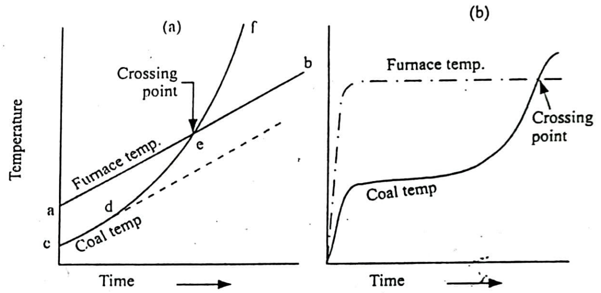


Figure 1.2 Determination of crossing point temperature coal: (a) with increasing furnace/bath temperature (b) with constant furnace/bath temperature.

Crossing-point temperature method

In this method, the coal sample (4 g) is heated in a reaction tube in a furnace or bath at constant or rising temperature, with oxygen or air passing through it at a predetermined rate till the coal temperature crosses the furnace or bath temperature (Fig. 1.2). The temperature at which the temperature of the coal and the furnace/bath coincide is called the crossing point temperature. It is also called the relative ignition temperature. The heating rate, the air-flow rate, and the particle size must be optimized.

Figure 1.3 gives a schematic diagram of the crossing point apparatus used by Raju et al. [10] with manual and automatic control using a temperature programmer for their studies on crossing-point temperatures of Indian coals.

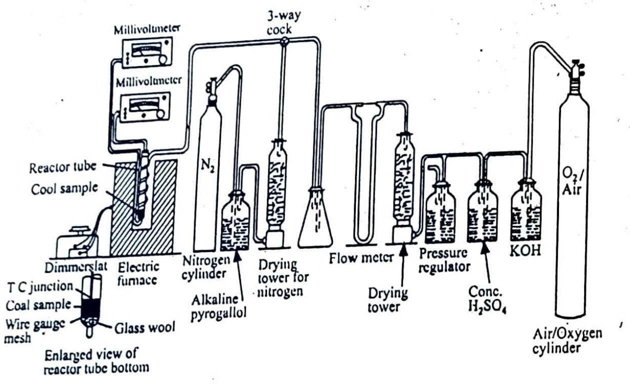


Figure 1.3 Schematic diagram of crossing point temperature apparatus.

Some researchers consider the point *d* in Fig. 1.2(a) to be more important than the crossing point for the simple reason that the temperature of the sample can overtake the furnace or bath temperature only after the process of perceptible self-heating has set in without additional heat. The heating curve can, therefore, be divided into three zones c–d, d–e, and e–f. The third zone starts from the crossing point to active combustion.

The crossing point temperature of Jharia coals lies between 189° and 231°C.

Differential thermal analysis (DTA)

Thermal analysis is a technique in which some physical or chemical changes of a substance are measured as a function of temperature as the substance is subjected to a controlled heating rate. Differential thermal analysis is one of the seven thermal analysis techniques used for the study of spontaneous heating of coals. In this technique, the difference in temperature (Δt) between a substance and a thermally-inert reference material is recorded against temperature as the two specimens are subjected to identical temperature changes in a block which is heated at a linear heating rate. If the sample undergoes no thermal change, and the temperature difference is zero, a so-called baseline parallel to the temperature axis (that is $\Delta T = 0$) is given by the recording instrument. However, if the sample undergoes a thermal decomposition reaction with loss or gain of heat, the temperature gradient with respect to the inert material will change, producing a negative or positive ΔT, depending on whether the transformation taking place in the sample is endothermic or exothermic. The curve recorded with ΔT plotted on the ordinate and temperature or time on the abscissa is called the DTA curve or thermogram.

The DTA method is being increasingly applied over the last three decades for the study of coals and coal products.

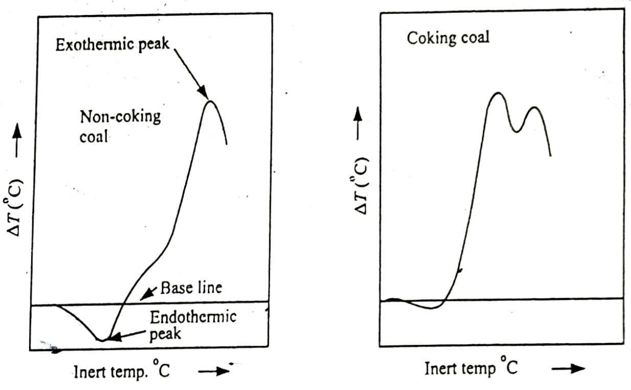


Figure 1.4 Typical thermograms for coals.

Typical thermograms for coal are shown in Fig. 1.4. There is first the endothermic region due to evaporation of water with the curve going below the baseline and subsequently returning to the baseline after an endothermic peak. As the temperature rises due to oxidation, the curve rises steeply. The steeper the rise of the curve, the faster is the heat evolution. A slow rise after the endothermic region indicates a higher ignition temperature. The curve then attains a single peak and falls, unlike with coking coals, in which two peaks with a valley in between are observed. It is known that low-rank coals have lower ignition temperatures and higher susceptibility to spontaneous heating compared to higher-rank coals. A comparison of DTA curves would help in determining the relative liability to spontaneous heating of coals.

1.1.1.2.6 Relative liability of coals to spontaneous heating

Although the occurrence of a fire in a mine due to spontaneous heating of coal is the result of the interplay of a large number of factors, a knowledge of the relative liability of coal seams would be useful in making important decisions when planning mining operations in a new seam. There is no single accepted method of evaluating the liability of a coal to spontaneous combustion. There are several methods which utilize correlations existing between known properties or characteristics of coal and susceptibility to spontaneous combustion. Different methods have thrown out various indices, the comparison of which would, however, be inaccurate and possibly misleading owing to the diversity of experimental conditions. For a laboratory comparison of the relative heating tendencies, only one method may be used under standard conditions, and adiabatic calorimetry is the preferred choice, although it is time-consuming and laborious. The DTA technique may also be used for fast, accurate, and reproducible results. The crossing point method offers a cheap, though not quite accurate method of determining the crossing point temperatures. In many instances, past experience with the working of given seam(s) in other mines and storage of coal on the surface is the most commonly used indicator of hazard due to spontaneous heating.

The most common indices that may be, used are the following :

Heating rate It is the temperature increase under controlled conditions in °C/h g

Heat release It is the quantity of heat released per gram (ΔH) in cal/g.

Crossing-point temperature It is the temperature value at which the temperature of coal coincides with that of the heating furnace or bath in °C.

Ignition temperature It is the minimum temperature at which the coal ignites without an additional external source of heat on account of the exothermic reaction with oxygen. It is determined by observation.

Oxidation rate It is the consumption of the oxygen given in mg per kg coal per minute, (mg O_2 / kg min.) or in mg per g coal per hour, (mg O_2 / g h).

Adiabatic self-heating time It is the time interval between the starting temperature of the experiment and ignition of coal.

Ignitibility or liability index with chemical method
It is the difference in puff or 'explosion' temperatures of coal in the fully-reduced and fully-oxidized states.

Liability index proposed by Feng et al.[11]

$$\text{Index} = \frac{\text{average heating rate between } 100^\circ \text{ and } 220^\circ\text{C}}{\text{relative ignition temperature}} \times 100$$

This is based on the assumption that the average heating rate between 110° and 220°C is only slightly influenced by the evaporation of moisture and the exit of volatile matter from the coal in this temperature range. Coals were classified as follows:

Index	Liability to spontaneous combustion
0-5	Low
5-10	Medium
> 10	High

Liability indices proposed by Mahadevan and Ramlu [12]
Susceptibility potential using the chemical method is defined as the difference in puff temperatures of coal in fully-reduced and as-mined/as-received states ($T_R - T$) the extreme limit of which is $T_R - T_O$, where T_O is the puff temperature in fully-oxidized state.

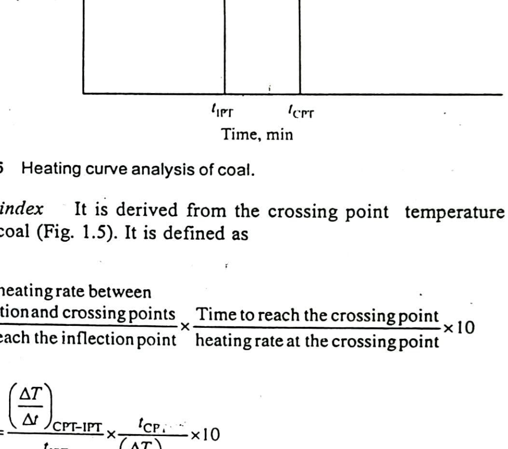


Figure 1.5 Heating curve analysis of coal.

Liability index It is derived from the crossing point temperature heating curve of coal (Fig. 1.5). It is defined as

$$\frac{\text{Average heating rate between the inflection and crossing points}}{\text{Time to reach the inflection point}} \times \frac{\text{Time to reach the crossing point}}{\text{heating rate at the crossing point}} \times 10$$

$$= \frac{\left(\frac{\Delta T}{\Delta t}\right)_{\mathrm{CPT-IPT}}}{t_{\mathrm{IPT}}} \times \frac{t_{\mathrm{CP}}}{\left(\frac{\Delta T}{\Delta t}\right)_{\mathrm{CPT}}} \times 10$$

This index which increases with the increasing tendency of coal to spontaneous combustion takes into account the entire heat-up history of the coal right from the ambient temperature. It is only necessary that the experiments are conducted under standard conditions.

1.1.1.2.7 Fire-risk rating of coal mines due to spontaneous combustion

Attempts were made by some research workers to evaluate the fire risk rating of coal mines on the basis of the possible fire hazard from working of seams liable spontaneous combustion. Feng et al. [11] proposed a method in which a risk index is obtained for a mine as a product of the liability index as determined by them from the crossing-point temperature heating curve for coal, the mine environmental index estimated as a function of coal loss in extraction, fissuration of the coal mass and strata as a result of mining, and ventilating pressure differentials all of which are governed by geological and technical factors.

Risk index = Liability index × Mine environmental index

Table 1. 1 gives the various indices and ratings proposed which may help in categorizing different coal mines.

Banerjee [13] suggested a methodology for computing the total fire-risk inherent in a particular situation obtained in a mine taking into account the amount of risk involved in each of the different mining operations being carried out under different conditions in the mine. Mahadevan and Ramlu [12] utilized the method of Feng et al. to estimate the mine environment index based on Indian conditions. They, however, used a new liability index derived by them from the crossing point temperature heating curve.

Table 1.1 Liability index-Environmental index–Risk index

<table>
<tr><th colspan="3">Liability index for coal</th><th colspan="2">Rating</th></tr>
<tr><td colspan="3">0–5</td><td colspan="2">Low</td></tr>
<tr><td colspan="3">5–10</td><td colspan="2">Medium</td></tr>
<tr><td colspan="3">>10</td><td colspan="2">High</td></tr>
<tr><th colspan="5">Mine environmental index</th></tr>
<tr><th>Coal loss</th><th>Fissuration</th><th>Ventilation pressure differential</th><th>Environmental index</th><th>Rating</th></tr>
<tr><td>Normal</td><td>Natural</td><td>Normal</td><td>1</td><td>Low</td></tr>
<tr><td>High</td><td>Natural</td><td>Normal</td><td></td><td></td></tr>
<tr><td>Normal</td><td>High/induced</td><td>Normal</td><td>2</td><td>Medium</td></tr>
<tr><td>Normal</td><td>Natural</td><td>High</td><td></td><td></td></tr>
<tr><td>Normal</td><td>High/induced</td><td>High</td><td></td><td></td></tr>
<tr><td>High</td><td>Natural</td><td>High</td><td>3</td><td>High</td></tr>
<tr><td>High</td><td>High/induced</td><td>Normal</td><td></td><td></td></tr>
<tr><td>High</td><td>High/induced</td><td>High</td><td>4</td><td>Very high</td></tr>
<tr><th colspan="5">Risk index for mine
(Liability index × Mine environment index)</th></tr>
<tr><th colspan="3">Index</th><th colspan="2">Rating</th></tr>
<tr><td colspan="3">0–1</td><td colspan="2">Low</td></tr>
<tr><td colspan="3">10–20</td><td colspan="2">Medium</td></tr>
<tr><td colspan="3">20–40</td><td colspan="2">High</td></tr>
</table>

1.1.1.2.8 Detection of spontaneous combustion

Early detection of spontaneous combustion is of utmost importance as experience has shown that late discovery of a heating may prove to be very costly to the mine.

The spontaneous combustion of coal in mine workings may be detected by the physical and chemical indications given by the heating itself.

Physical indications

The traditional physical examination for physical indications consists in detecting the presence of warmth, sweating, and smell of fire in the district return airway or in the path of air coming from a suspected area. In practice, the first sign that a heating has developed is the characteristic smell of 'fire stink' or 'gob stink' due to the gases distilled from coal and products of sulphur if the coal contains a high percentage of sulphur. In the initial stages of heating, the smell resembles that of petrol, but as the heating progresses, it changes to the odour of kerosene, and later to the odour of tar. A strong smell is often an indication that the heating has made sufficient headway. In mines, it is difficult to find personnel who are experienced in the detection of a heating by smell. Sniffing should be restricted to the general body of air currents and not be practised with gases issuing from gateside packs of longwall faces and cracks in the strata. The smell may not be detected where oxidation products have to travel long distances through caved rock in the goaf before reaching an airway or where a heating is deep-seated or is near an intake airway. Although, in practice, most fires are first detected by mine personnel by seeing and smelling smoke or seeing the fire start, too much reliance should not be placed on it. An air analysis is always more reliable to provide the first warning.

Chemical indications

As coal passes through different stages of spontaneous combustion, gaseous products such as CO, CO_2, CH_4, C_2H_6, C_nH_{2n}, and H_2O are liberated, the composition of the gaseous products depending on the stage of oxidation. CO and CO_2 are evolved at low ambient temperatures up to 100°C. Whereas CO is produced in measurable quantities even at temperatures of 30°C, the concentration of the hydrocarbons at temperatures not greatly above the ambient cannot be measured by the present-day sampling techniques. As the concentration of CO increases consistently with temperature during the low-temperature stage of the oxidation process, it has been used as the most sensitive indicator of the onset as well as progress of the spontaneous combustion of coal. CO_2 may be produced from other sources or may be lost due to sorption by coal or by solution in water. As CO may be produced from shotfiring or from the use of diesel haulage equipment, its concentration must be determined at carefully selected sampling points at times when contamination of air will not take place from these sources. CO entering the mine from the surface along with the downcast air due to nearness of the downcast shafts to the boiler house, carbonisation plants, etc. cannot, of course, be eliminated.

To detect any increase in CO content, one must, therefore, know the, content normal for a district or a seam under given conditions so that any increase above it may be viewed with suspicion and immediately investigated.

Today, computerized CO monitoring and control systems have been adopted for early detection of spontaneous combustion in several coal mines throughout the world.

Graham developed in the 1920s, a ratio called the Graham index or Graham ratio, given as the ratio between the actual concentration of CO produced and the deficiency of O_2 compared with fresh air. This ratio which is also called the carbon monoxide/oxygen deficiency ratio or carbon monoxide index, has stood the test of time and is extensively used throughout the world for predicting the presence and development of spontaneous heating. It is usually multiplied by 100 to give a convenient number.

With a natural oxygen–nitrogen ratio of 0.265 in intake air,

$$\text{CO Index} = \frac{100\,CO}{0.265N_2 - O_2}$$

$$N_2 = 100 - CH_4 - CO - CO_2 - O_2 - H_2$$

where CH_4, CO, CO_2, H_2, and O_2 are the percentages of the gases present at any given time in a sample of air coming from a suspected area in a mine. Mine air may contain small traces of CO up to 0.005 per cent even when there is no fire, and as CO may also enter the mine from the surface, the background $CO/\Delta O_2$ ratio in the return air of every district must be established.

$$\text{CO Index} = \frac{\text{final CO} - \text{initial CO}}{\text{initial } O_2 - \text{final } O_2}$$

The normal value of the CO Index is usually 0.4 or less. In German hard coal mines, values up to 30 with incipient fires and up to 90 with open fires have been obtained [14].

When an inert gas is introduced into a fire area without closing the ventilation tunnel or pipe in the stopping on the return side,

$$\text{CO Index} = \frac{100\,CO}{\left(\frac{O'_2}{N'_2}\right)N_2 - O_2}$$

where O'_2 and N'_2 are the percentages of the gases in the intake air samples and O_2 and N_2 the percentages in the return air samples.

One should, however, bear in mind that the CO Index is reduced by dilution with blackdamp and that the ratio can be used only when ΔO_2 is not less than 0.5per cent by volume due to an accuracy of ± 0. 1 per cent that is obtained in measuring the oxygen concentration. Also, an error in the calculation of the index increases with a decrease in CO and an increase in O_2

Monitoring mine air for CO and O_2 will enable determination of the CO Index.

1.1.1.2.9 Prevention of spontaneous combustion of coal in coal mines

The best safeguards against mine fires due to spontaneous combustion consist in conducting mine operations carefully and systematically. The following points should be observed when working seams liable to spontaneous combustion:

Development

(1) Main shafts should be properly located, lined, and equipped so that there is less pressure loss.

(2) Development within the coal seam should be kept to a minimum. All coal surfaces not being directly worked on must be protected against absorption of oxygen by stonedusting, guniting, or coating with an antipyrogene (a fire-proofing composition).

(3) When working seams by the bord and pillar method:
   (a) the dimensions and shape of the pillars must be such that no crushing of the pillars takes place before or during their extraction which will increase the air permeability of the fractured coal;
   (b) panels with independent ventilation should be formed as far as practicable;
   (c) the panels must be laid so that severe crushing of coal during extraction is minimized;
   (d) the panels formed must be of a size which would permit complete extraction within the incubation period; and
   (e) the development work should be kept out of abutment areas.

(4) When designing longwall face layouts, face lengths should be selected so that good closure and compaction of the goaf are assured. Face lengths smaller than 60 to 65 m are generally not recommended.

(5) The roof of roadways in coal should be well-supported and the sides (ribs) well-lagged.

(6) When traversing seams with cross-cuts, coal should be removed to an adequate depth and the cavities so formed should be filled or packed with clay, pulverized ash with portland cement, or other incombustible material.

(7) Side-by-side intakes and returns must be avoided as far as practicable. The main intake and return airways and the main haulage roadways should be driven in stone as far as practicable. In thick and inclined seams, the main roadways should be driven in the floor of the seams with cross cuts driven to the seams.

(8) The maximum air travel distance from surface to surface (which ranges from 1 to 20 km) must be kept low so that the number of stoppings and the leakage that takes place are reduced.

Coal winning

(1) The coal should be won as completely as practicable using an efficient extraction method, especially in disturbed zones, thick seams, and strongly folded steep seams.

(2) The working faces should be worked intensively as a high rate of extraction is the best means of preventing fires due to spontaneous combustion. The advance or retreat of a coal face should never be interrupted and, if any local fall of roof or a fault is encountered, immediate steps should be taken to get over the obstacle or minimize its effect on the face advance.

(3) When mining a group of seams or mining thick seams by longwall slicing, the workings in an upper seam or slice should be in advance of those in a lower seam or slice and the coal of the upper seam or slice should be extracted as completely as practicable.

(4) The best methods of working seams are the longwall and stall methods.

(5) When working a seam by longwall mining, the retreating longwall should be preferred as it eliminates stray leakage currents through the goaf behind the face if regular caving of overlying strata behind the face is accomplished. Retreating is the best method if there is a goaf above the seam. It is the only method for working thick inclined seams by sub-level caving with inertization of the goaves behind faces using liquid nitrogen or nitrogen foam to combat the high spontaneous combustion risk.

(6) As it is not always possible to avoid coal remains in the goaves of longwall faces even when powered supports are used for extracting a seam by longwall advancing with caving, the gateside packs should be airtight in order to increase the airflow resistance of the goaf; also one must balance the pressure on all sides of the goaf. When extracting a seam by retreating longwall, the tail gate and the main gate behind the face line must be stowed as tight as possible or they should be sealed off at regular intervals. When starting a new face from solid, the gate pack must be well tied into the barrier pillar

(7) Worked-out areas should be effectively sealed off as soon as possible or ventilated. Stowed seals should be preferred to other forms of seals. If the worked-out areas are ventilated, as they are in most American coal mines, a system of bleeder entries must be planned.

(8) Blasting in coal must be restricted as much as possible.

(9) Loose coal and other combustible material, such as oily or greasy waste and decaying timber, should cleaned up and not be permitted to accumulate.

(10) In bord and pillar workings, the pillars should be mined as fast as practicable at an even rate after their formation.

(11) Partially-built seals or preparatory stoppings should be erected in the entries of panel fire barriers before extraction is commenced.

(12) When extracting folded seams, the limbs of a synclinal or anticlinal fold must be extracted, as far as possible, simultaneously and uniformly with stowing.

(13) When working seam outcrops with shallow overburden, barrier pillars of sufficient size should be left to the surface so that air leakage through fissures is avoided.

Roof control

(1) Solid stowing is the best method of roof control for working seams liable to spontaneous heating. Pneumatic and hydraulic stowing are eminently suitable as stowing methods.

(2) Where advancing longwall mining with caving is practised, the gate roads should be supported with tight roadside packs.

(3) In bord and pillar workings with open caving of goaf, a straight line of fracture should be maintained. The fracture line should be oriented to the natural fracture system of the roof so that caving is promoted and long roof spans projecting over the goaves are prevented. The extraction must also be planned so that points formed by converging fracture lines are avoided as far as is practicable.

(4) Oxidizable waste materials, native or foreign, should not be discarded into the goaves.

(5) Washery dirt, if used for stowing, should be admixed with crushed rock so that, when in place, it does not constitute a fire risk.

Ventilation

(1) All active mine workings must be adequately ventilated.

(2) The mine ventilating pressure should not be unduly large so that air leakages through crushed roadway pillars or defective stoppings or seals of sealed-off areas do not take place.

(3) Short-circuiting of air as well as its uncontrolled circulation must be eliminated. Systematic air leakage tests should be conducted in workings of high-risk seams by quantity surveys and smoke tube tests.

(4) All roadways in coal should be ventilated and unused roadways sealed off.

(5) Roadways in coal should not be wide and high, so that the air currents flowing through them have sufficient velocities to facilitate heat dissipation.

(6) Ventilation stoppings, doors, and regulators should be correctly placed.

(7) Air-crossings should be airtight and be constructed of fireproof material.

(8) Mine ventilation should be efficiently supervised. The ventilating air current in the return of each ventilating district where coal is liable to spontaneous combustion must be checked for CO concentration at least once per day.

Inspections

(1) Every underground working place should be inspected by a supervisory official or competent person at least once during each working shift. During inspections, attention should be paid to the presence of and means of eliminating fire risks arising from spontaneous combustion and other causes.

(2) On idle days, all districts liable to spontaneous combustion should be inspected at least once by competent persons.

1.1.1.3 Electricity

The advantageous electrification of mines has brought in its wake numerous hazards of short-circuiting and sparking due to improper methods of installation, use, repair, and maintenance of electrical equipment most of which are constantly on the move as mining advances or retreats. Short-circuits and sparks may ignite any combustible material present in the immediate vicinity if proper safeguards and preventive measures are not installed.

1.1.1.4 Friction

Fires due to frictional causes occur in roadways and inclined shafts/inclines in which rope haulages and belt conveyors are installed; in staple shafts which are supported with timber; and at coal faces which are cut by coal-cutting machines, continuous miners, or shearer leaders. Belt conveyor fires form the largest percentage of these fires.

Belt conveyor fires

Nearly all belt conveyor fires result from frictional heating. The chief causes are:

(1) Overheating at the drive pulleys due to their continuing to rotate against a stalled or slipping belt;
(2) Overheating at the carrying and return idlers with rolls rotating against obstructions; and
(3) Overheating of brakes and gears.

Fires associated with belt conveyors can be minimized by adopting the following preventive measures:

(1) Using fire-resistant belting of neoprene (a trade name for polychloroprene), PVC (polyvinyl chloride), and styrene butadiene;
(2) Using fireproof roadway support at the drive and tail-end units;
(3) Selecting suitable electrical equipment and installing it properly;
(4) Installing the conveyor so that there are adequate clearances between it and the roadway floor and sides;
(5) Installing an automatic fire sensor system at the beginning and end of each belt flight, at the belt drive, and along the belt flight that will stop the drive automatically should dangerous overheating develop and provide an automatic warning, both audible and visual. It may be used to actuate a deluge-type water system, foam generator system, multipurpose dry powder system, or other equivalent automatic fire suppression system. Of the various types of fire detectors successfully developed and used for mine conveyor belt applications, the thermal point-type detector using a thermistor sensor has been found to be adequately sensitive for early fire detection, rugged to withstand the mine environment, simple and economical to use, and not actuated by combustion products from areas other than belt haulage roads. The point-type sensors should be closely spaced in the roadway as the hot gases from the fire cool rapidly when mixed with the ventilating current;
(6). Installing, safety devices that will automatically stop the conveyor drive should the belt break, jam, slip excessively, or fall below its normal speed;

(7) Preventing accumulation of dangerous electrostatic charges;
(8) Inspecting at frequent intervals while conveyors are working and at suitable times on idle days;
(9) Maintaining correct alignment and tension of the belt;
(10) Checking constantly the running condition of the idler rolls and bearings;
(11) Keeping the idler rolls clear of coal dust and all obstructions that might cause friction;
(12) Repairing the belt promptly and preventing exposure of the belt carcass;
(13) Making and maintaining belt joints as tight as possible;
(14) Clearing, at frequent intervals, coal falling on the cover plates and on the roadway floor under the belt;
(15) Cleaning transfer points at regular intervals;
(16) Providing a good communication system that will provide contact along the main points on the conveyor system and other parts of the mine;
(17) Providing fire protection by installing water lines along belt lines with fire hose outlets at 100-m intervals, multi-purpose dry powder extinguishers, automatic sprinklers, or automatic foam generators at all drives and electrical controls;
(18) The roadway support at conveyor drives and transfer points should be fireproof; and
(19) Providing proper ventilation.

Staple shaft fires

Fires in staple shafts by friction may occur through an unattended hoist starting up of its own accord while its brakes,are in the 'On' position. Frictional sparks may be produced at the brake rim which may ignite coal dust settling upon it. Where a friction hoist is used, hot abraded particles of the rope tread if of aluminium alloy, may ignite a pile of fine coal dust immediately beneath and the fire may spread to the support of the engine room and finally to the shaft. Fires in staple shafts may be prevented by using brake lining and rope tread of fire-resistant materials, constructing fireproof engine rooms, and providing fire protection in engine rooms and shafts by means of an automatic sprinkler system.

Fires in rope-haulage roadways and inclined shafts

Fires in rope-haulage roadways and inclined shafts may result from haulage ropes rubbing on frozen rollers or coming into contact with surfaces of combustible materials such as coal and timber, and from overheated bearings of mine cars. They may be prevented by proper selection, installation, and maintenance of rollers, pulleys, and car axle bearings. The shaft rollers may be rubber-faced. Coal or other carbonaceous materials should not be utilised as packing material in track-laying.

Fires in machine-cut and power-loaded faces

Fires may occur on machine-cut dry coal faces from frictional sparking, during orthodox coal-cutting operations in thin seams which have a hard sandstone roof or floor or intrusions of quartzitic or pyrite rock at the cutting horizon. The frictional sparks produced by the travelling picks can ignite a flammable mixture of firedamp or dust, causing burning or an explosion which may start a fire (Fig. 1.6). The most effective method known at present to prevent formation of flammable mixtures near the cutting picks is to provide additional ventilation in the cut to dilute the mixtures below the lower flammable limit.

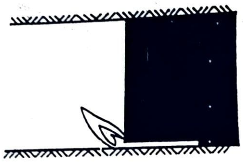


Figure 1.6 Fire in undercut.

Fires may occur on power-loaded faces from frictional ignition of firedamp during the coal-cutting operation. The firedamp may burn into or spread along, the surface of the small coal which is left in the track behind the machine and through which firedamp filters at a rate which will be maximum immediately after the coal has been cut [15].

1.1.1.5 Blasting

Blown-out shots in coal especially when coal dust is present in shotholes may cause mine fires through ejected glowing particles igniting coal dust or loose coal near the face. Inducer shotfiring in solid coal when working seams subject to coal and gas outbursts creates conditions for spontaneous heating of coal despite its low liability to spontaneous combustion. Blasting can also ignite an unknown methane blower which in turn may set an entire coal face on fire. Blasting is definitely a minor cause of fire.

1.1.1.6. Explosions

Firedamp and coal-dust explosions leave in their wake smouldering and active fires. The explosion flame, explosion gases, and coal dust taking part in an explosion may set on fire wooden doors, paper, non-fireproof brattice line, pieces of clothing, unburnt coal dust and loose coal, and ultimately coal faces. The danger of fires from explosions increases in the colder months.

1.1.1.7. Miscellaneous

Welding, cutting, and soldering operations with an arc or open flame underground may result in mine fires. As a precaution against such fires, these operations should be performed only by competent persons under the constant supervision of a qualified person at places free from danger due to firedamp or coal dust and where adequate fire-extinguishing appliances and stone dust are immediately available.

Methane blowers may catch fire and the gas flame may start a fire.

Static or frictional electricity may also cause fires indirectly from flammable firedamp–air or coal dust–air mixtures.

Friction of belts against pulleys and passing of dust-laden air through ducting may generate charges sufficient to produce sparks. The danger can be greatly minimised by installing all machinery so as to secure a safe discharge of static electricity.

Diesel-powered equipment, if not maintained properly, may become a cause of fire due to the risk of malfunctioning of the equipment. Diesel oil can also feed a fire. Their maintenance facilities should, wherever possible, be located in return airways.

Fires following coal-dust explosions resulting from deephole blasting in coal may occur when opencasting shallow seams which were earlier laid out on the bord and pillar system for underground mining.

Glow fires may originate on electric face lights and on excessively heated surfaces of electrical equipment due to ignition of fine coal dust deposited on them by undissipated heat of radiation.

Fires may occur on compressors which can be controlled through proper maintenance and inspections.

Surface fires may communicate with underground workings.

1.1.2 Detection of open fires

For the detection of open fires which, in general, evolve much larger quantities of CO than incipient fires, CO recorders can be installed in main return airways and at upcast shafts on the surface.

1.1.3 Localization of mine fires

Mine fires can be localized or confined to the place of their origin. Localization may be effected by using fireproof or fire-resistant building or construction materials for the support of mine workings or for fighting a fire either directly by fire-extinguishing means or quickly controlling the air supply to it.

Roadway fires can be localized by one or more of the following measures:

(1) Forming fire-resistant or fire-arresting zones of incombustible support at least 75 m long at distances of 500 m from one another in the case of long roadways or less in the case of short ones;
(2) Forming fire-protected zones by applying a fire-proofing coating of calcium chloride, calcium chloride-clay, sodium silicate-clay, vermiculite-portland cement, magnesium-oxysulphate cement, steel fibre-reinforced concrete, gypsum-fibre glass, and foamed plastic (Isoschaum) on roadway sides, roof, and floor. Polyurethane foam should not be used as it burns away at temperature of 150°C producing a thick black smoke containing ammonia, poisonous cyanate, amines, carbon dioxide, carbon monoxide, and other complex gases;
(3) Installing automatic sprinkler systems which will form deluge-type water spray barriers about 10 m long (Fig. 1.7) to cool hot gases from the fire and prevent the latter from spreading;
(4) Lining roadway crossings and junctions with fireproof material such as brick or concrete especially in mines where a considerable fire hazard exists;
(5) Constructing fire-resistant fire doors in the pitbottoms of all shafts at all working levels so as to isolate shafts from the levels in case of fire.

Staple shaft fires can be localized by the following measures:

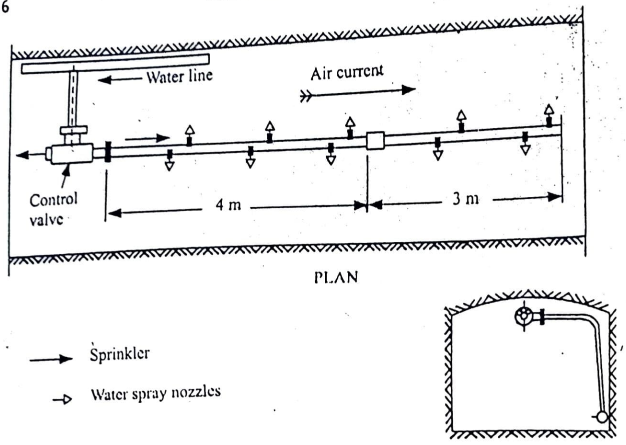


Figure 1.7 Automatic sprinkler system in roadway.

(1) Sinking the staple shafts a little away from the cross-cuts and laterals in flat measures laid out on the horizon mining system so that the shafts may be reliably cut off from air currents in the roadways in the event of fire;
(2) Using fireproof shaft lining;
(3). Constructing the engine rooms and tops of shafts fireproof; and
(4) Providing steel fire doors at the bottom, top, and any intermediate landings.

Fires in substations, rectifier rooms, battery-charging stations, oil storage rooms or chambers, pump chambers, and repair and maintenance shops can be localized by constructing them fireproof and equipping them with automatic fire doors and firefighting equipment.

Where compressed-air supply is available, the compressed air may be used to build up pressure of fresh air inside the hoist and pump chambers, in addition to protection by fire doors, to counteract entry of fire gases to a large extent.

Fires due to spontaneous combustion can be localized and extinguished in the incipient stage by direct attack with water, sand, stone dust, or chemicals, but when they have gained such headway that it is impossible to fight them directly, they require to be cut off from air reaching them.

1.1.4 Fighting mine fires

The common methods of fighting mine fires are:

(i) Fighting by direct attack;
(ii) Fighting by indirect attack –

(a) Isolation of the fire;
(b) Sealing the fire area or the entire mine;
(iii) Flooding the fire area or the entire mine;
(iv) Flushing the fire area with sand or other suitable solid materials conveyed with water;
(v) Introducing an inert gas into the fire area; and
(vi) Special methods of firefighting.

There can be no fixed rules for the application of one or the other method for dealing with mine fires as each fire presents a problem of its own which requires careful thinking. The position, intensity and extent of a fire, depth and layout of the workings, degree of gassiness of the mine, and immediate availability of firefighting facilities govern the selection of a method. In any method of extinguishing a fire, safety should be given the utmost consideration. However, it frequently happens in mines that, in order to prevent loss of production or damage to property, improper methods of firefighting, at great risk to life, are followed. Fighting fires in confined spaces may be attended with serious consequences.

1.1.4.1 Fighting by direct attack

Many mine fires, especially the open ones, can be checked and extinguished in their early stages by direct attack if they can be reached by digging out the heated or burning material into cars for sending out of the mine after covering it with stone dust or by attacking them with water, sand, stone dust, chemicals, or other smothering materials. Failure to extinguish fires by direct attack is usually due to late discovery, lack of firefighting means, or following improper firefighting procedures, so that the fire gets beyond control and a chain of events may follow that may result in extensive damage to property and loss of life.

When digging out a fire pocket, the resulting cavity should be immediately filled with an incombustible material such as clay, screened boiler ash mixed with porland cement, broken rock or brickwork, or an approved sealant. This method is less frequently used due to the probable risk of ignition of firedamp, spreading of the fire while transporting the hot material outbye, or increasing the intensity of the fire by providing free access of air to it.

When fighting fires directly, the quantity of air reaching the fire must be controlled so that the fire does not intensify or propagate unnecessarily in the direction of the air current. With ascensional ventilation, an increase in the ventilating air occurs due to natural ventilation as a result of fire, which may cause reversal of air current in a parallel airway, while with descensional ventilation a decrease in ventilating air or reversal of air current may take place. The air quantity should, however, be controlled to be sufficient enough to prevent backing up of smoke and gases against the air current due to convection effects (Fig. 1.8) or reversal of air current. The backing up of smoke and gases against the ventilating current seriously affects the visibility and access to the fire. The smoke and gases backing out from a fire in an airway flow adjacent to the roof of the airway has a thickness which is related to the air velocity, the temperature.

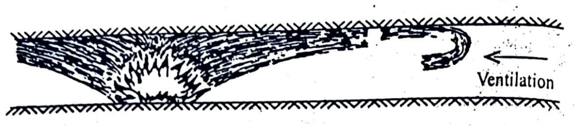


Figure 1.8 Backing up of firegases.

and heat radiated from the fire. Control of air current reaching a fire is not easy as it may affect the safe working of the rest of the mine, from which samples of air must be continuously collected for evaluation.

To obtain access to a fire where the smoke and gases have already backed out from it, it is necessary to use hurdle cloths which will cause the incoming intake air current to be carried up to the roof level, where it meets the warm air and gases tending to flow outbye, thereby cooling them and causing them, under the ventilating pressure, to flow back in the normal direction of the ventilation. As the smoke and gases are cleared in the immediate vicinity of the hurdle cloth, the latter can be shifted forward to clear the roadway, section by section, until the fire is accessible. The height at which a hurdle cloth is effective must be found by trial. It should be such that it will clear the smoke but not reduce the ventilation.

Fundamentally, a fire should be attacked from the fresh-air side in the direction of its propagation and not against it. If a fire cannot be extinguished within a few hours of starting firefighting operations, it would be advisable to resort to some other method, sealing off in all probability, as prolongation of the firefighting efforts might prove disastrous.

1.1.4.1.1 Firefighting facilities

Every mine should be provided with suitable and adequate firefighting facilities so that different classes of fires (fires from solid combustible materials, fires from flammable liquids, and fires from energized electrical equipment) may be quickly controlled. The minimum firefighting facilities that should be made available at a mine depend on the hazardous character of the mine, number of men employed, daily output, and size of the mine.

The standard firefighting facilities which are considered when planning fire protection in mines are:

(a) Water mains or water lines;
(b) Sprinkler systems;
(c) Truck-mounted water tanks;
(d) Fire extinguishers;
(e) Stone dust, sand, or dry chemical; and
(f) Miscellaneous firefighting materials and tools.

1.1.4.1.1.1 *Water mains*

Water, without any doubt, is an excellent fire extinguishing agent for fire at any stage. It can be used as a solid stream or as a spray, depending on the nature and location of the fire. It usually works by cooling the surface of the burning material. The only conditions for its efficacy are that it should be available in sufficient quantity and that adequate means for distribution are available. The water mains should be capable of giving at least 400 litres per minute of water at a minimum flowing pressure of 1.5 bar(g). They should preferably be connected, where practicable, to a covered steel or reinforced concrete static water reservoir/tank on the surface (Fig. 1.9) rather than to the conventional public water system. The tank should have a storage capacity of at least 100 m^3 and should preferably not be used for any other purposes. In mines which are located so that a constant adequate supply of fresh water is not available from streams, lakes or ponds, water stored in underground sumps or suitably worked-out areas may constitute a supply for firefighting. In such cases, connections, bypasses and valves must be installed on the discharge line at the pumping station to supply

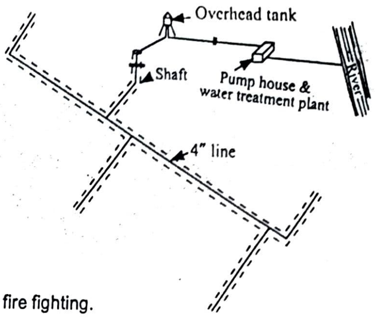


Figure 1.9 Water supply for fire fighting.

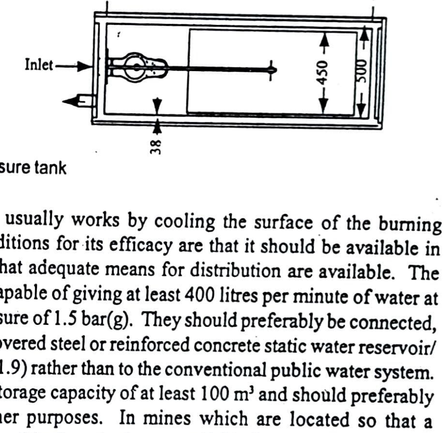


Figure 1.10 Break-pressure tank

water under sufficient pressure to the working sections along the main and panel haulage roadways. In deep mines, in order to prevent excessive pressures at hydrants or hose taps, cascade dams or break-pressure tanks with ball float valves or pressure-reducing valves are provided at different insets (Fig. 1. 10).

Water mains should be provided at all mines, not naturally wet throughout, at the top and bottom of the main shafts, in main and secondary haulage roadways, and in staple shafts. They should extend to all working sections of a mine with section control and isolating valves. They should be at least 50 mm in diameter (usually 100 mm) and be fitted with hydrants near the pitbottoms and at intervals not exceeding 100 m in the main roadways, 50 m in gate roads, 25 m in conveyor haulage roadways, and 20 to 25 m on the intake side of main junctions, engine rooms, fan houses, and substations, and at suitable central points in bord and pillar workings. The hydrants should be installed in such a way that they are protected against damage and are easily accessible and clearly visible. To prevent bursting of a fire-hose where the pressure at a hydrant exceeds 7 bar(g), a reducing valve or a pressure relief valve in the hydrant control valve should be used. The fire hose should be at least 38 mm (1.5 inches) in diameter (usually 2.5 inches or 2.75 inches for use from pressure pumps or water mains) and be quickly available in suitable lengths in coils stored in containers, recesses, or compartments especially provided for that purpose, or stored on reels at suitably located fire-hose stations. It should be fitted with instantaneous couplings with male inlet and female outlet. To obtain effective streams for firefighting purposes, the proper size and type of nozzle or nozzles should be selected for use with the standard size of hose used. Owing to the restricted space in roadways and the desirability of using the minimum quantity of water necessary, nozzles of 13 mm (0.5 inch) and 16 mm (0.625 inch) are usually used. Adjustable-type nozzles can also be used but they must be provided with a positive stop so that they cannot be closed to an extent which will cause bursting of the hose. A combination adjustable fog-and-firestream type may also be advantageously used, which will produce either a firefighting stream or the heavy fog for approaching the fire. It is claimed that water under pressure in the form of fog will effectively control the backing out of smoke and fumes, will reduce the danger of hydrogen explosion or steam scalds, and will reduce the temperature to a level that will permit directing firefighting streams of water by adjusting the nozzle after the fire has been reached.

In mines using compressed air, the air pipes or lines must be interconnected through pressure-reducing valves where necessary so that they can be quickly converted into water mains in an emergency.

The laying out of an efficient water main system can at times be a difficult problem. Consideration must be given to the quantity of water required at the farthest possible inbye point in the mine and the method of mine development followed so that the contour of the pipe-range is below the hydraulic gradient and does not intersect it.

To ensure reliable functioning of the water supply system, tests and examinations should be made at regular intervals to ensure that the water-supply at all hydrants is adequate and hoses, couplings, nozzles, etc. are in operating condition. The quality of water should also be periodically tested to see if it gives rise to corrosion or incrustation.

The underground use of water as a fire extinguishing agent may be associated with certain hazards. The water may cause a damaging physical effect on materials which have been subjected to intense heat from fire. If a jet of it is directed full on to the centre of a deep-seated large open fire of coal or coal face instead of attacking the fire from its fringes towards the centre, it may bring about an explosion due to water gas. In deep mines, wet-bulb, temperatures may be raised to such an extent that working conditions may become unbearable.

1.1.4.1.1.2 *Sprinkler systems*

Where they can be applied, sprinkler systems, automatic or manually operated, are efficient fire-protection devices. Automatic sprinklar systems as fixed fire-extinguisher systems, are advantageous because of their economy, reliability, good suppression capability, and minimum maintenance. They are particularly suitable for controlling fires in belt conveyor roadways, in timbered main and staple shafts, and in underground hoisting engine rooms (Fig. 1.11). In timbered shafts, the sprinkler devices installed should sprinkle every square metre of shaft section and be controlled from the landings or insets and shaft top. The amount of water required is at least 500 litres per minute per square metre of cross-section for main shafts and 400 litres per minute per square metre cross-section in the case of staple shafts. Each water sprinkler system should be examined weekly and a functional test of the complete system should be conducted at least once a year.

The automatic sprinkler head usually used has a sealed frangible bulb operating element. The small bulb of special glass contains a low boiling point liquid which does not completely fill the bulb as a small air bubble is left entrapped in it. As the liquid expands due to heat, the bubble is compressed and finally absorbed by the liquid. As soon as the bubble disappears, the pressure rises rapidly and the glass bulb shatters at a temperature of about 55°C actuating a water control valve and the water-spray nozzles. The exact actuation temperature may, however, be regulated by adjusting the amount of liquid and the size of the bubble when the bulb is sealed.

Sprinkler heads may also be equipped with a fusible link that will actuate at a temperature of 74°C or higher. Sprinkler systems have the disadvantage that their installation, care, and maintenance are somewhat complicated.

The position of all water mains, hydrants, valves and fire stations underground should be shown on a map which should be kept at the mine office.

1.1.4.1.1.3 *Truck-mounted water tanks, portable water cars*

In mines not equipped with water mains or when used as auxiliary equipment in large mines having water mains, mobile truck-mounted water tanks or fire cars should be provided. A truck-mounted water tank should have sufficient capacity, at

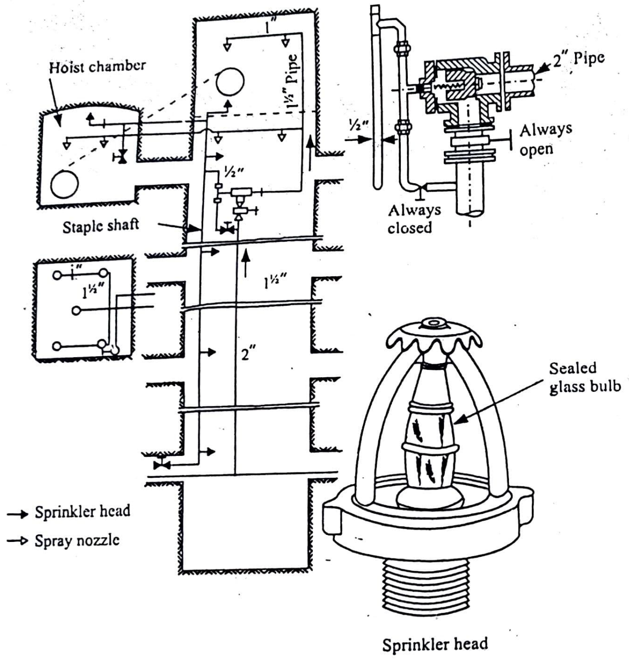


Figure 1.11 Automatic sprinkler system in staple shaft.

least 4 to 5 m^3, and be equipped with a pump or a compressed-air cylinder, fire hose, Siamese coupling, hydrant connection, and nozzles. The pump should be capable of delivering at least 200 litres of water per minute through a 50-m long, 38-mm diameter hose fitted with 13-mm nozzle at a nozzle pressure of 3.4 bar. The truck-mounted water tank should be stationed within 1500 m of each working section. In mines where such tanks are adopted as principal firefighting means, each unit should be stationed near the shaft-bottom, at intervals not exceeding 1500 m along the main and secondary haulage roadways, and within 800 m of each working section. However, the use of truck-mounted water tanks is limited in many mines as difficulties may be experienced in moving them from the place of storage to the place of a fire due to obstructions such as is offered by loaded and empty cars and other equipment. To increase the extinguishing ability of the water, sodium bicarbonate (8 per cent by weight) may be added in the water tanks.

1.1.4.1.1.4 *Portable fire extinguishers*

Most mine fires are small when they first start and they can often be easily controlled by the use of suitable hand-held portable fire extinguishers which are 'first-aid' firefighting appliances. When the fires assume larger proportions, they may still be fought successfully by large truck-mounted fire extinguishers or chemical firefighting trucks (chemical cars). When the gross weight of an extinguisher exceeds about 25 kg, it is usually placed on wheels.

Fire extinguishers are extremely handy when fighting fires at places which are not connected with water mains. They have greater extinguishing ability than water but suffer from a limited supply of the extinguishing agent. The maximum extinguishing effect is achieved when it is possible to cover or blanket the fire completely with the extinguishing medium. About 20 to 25 per cent of the workforce underground must be trained in the use of fire extinguishers.

The principles on which fire extinguishers suitable for use in mines function are:

(1) the cooling effect of water;
(2) the fire-extinguishing effect of carbon dioxide;
(3) the blanketing or shutting-off effect of foam;
(4) the smothering effect of solid substance; and
(5) the combination of more than one effect.

The smothering effect of halogenated hydrocarbons or Halon (Halon 1211 and Halon 1301) by dilution of the oxygen content of the air surrounding a fire and interfering with the chemical reactions in the fire zone is not utilised for firefighting underground because many of them are toxic and produce extremely toxic decomposition products at the elevated temperatures prevalent in fires.

The different types of hand fire extinguishers used in mines can be classified as follows:

(1) Soda-acid extinguishers;
(2) Foam extinguishers
    (a) Chemical foam extinguishers and
    (b) Mechanical or air-foam extinguishers;
(3) Carbon dioxide snow extinguishers;
(4) Dry chemical (dry powder) extinguishers; and
(6) Multipurpose dry chemical (dry powder) extinguishers.

Fire extinguishers are not all equally effective on different classes of fire. Only fire extinguishers certified by an approving authority, carrying appropriate labels as to size, type and rating and the classes of fires on which they can to be used, should be used underground. Hand fire extinguishers are generally designed with 5 to 10 kg capacity.

The various extinguishers described here are the common standard types. Basically, a fire extinguisher consists of a container, the extinguishing agent, a pressure-producing device or agent or the vapour of the liquid extinguishing agent itself, and a discharge orifice or a hose, and nozzle.

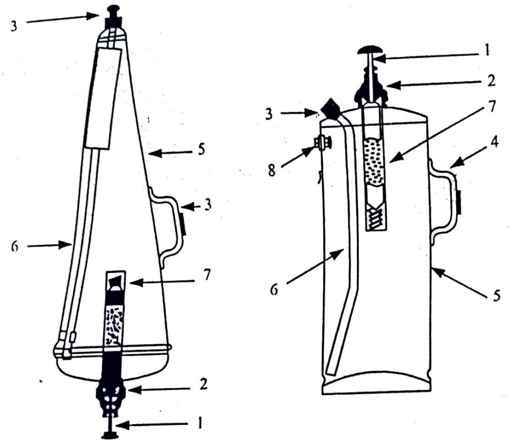


Figure 1.12 Minimax soda-acid exinguisher, 1: plunger knob; 2: plunger assembly; 3: nozzle; 4: handle; 5: container; 6: siphon tube; 7: metal cage with acid bottle; 8: safety valve.

Soda-acid extinguishers

This type of extinguisher depends for its action on the cooling effect of a jet of liquid. The extinguisher consists of a conical or cylindrical sheet steel container (Fig. 1.12) charged with basic sodium bicarbonate solution, a siphon or discharge tube, a nozzle, a hermetically-sealed acid phial or bottle, a perforated acid bottle metal cage, and a plunger knob assembly. The nozzle is closed with a protection disc which ruptures at a pressure of 2 bar(g) and has the object of preventing the stopping of the nozzle besides protecting the solution against the action of the surrounding atmosphere. The rupture disc, furthermore, builds up a pressure inside the extinguisher when the latter is operated, which is sufficient to force the extinguishing liquid out of the nozzle in a powerful jet after breaking the disc.

The extinguisher is operated by breaking the acid phial with the plunger knob, permitting the acid and soda solutions to mix, and directing the nozzle at the same time towards the fire with the extinguisher held almost horizontally. The chemical reaction produces carbon dioxide in such a quantity that a pressure of about 6.5 bar(g) is developed which forces the whole of the contents of the extinguisher to discharge on to the fire.

A 10-litre capacity soda-acid extinguisher has a horizontal covering range of up to 14 m and a discharge time of 90 seconds. Soda-acid fire extinguishers are suitable for fires of solid combustible materials. They cannot be used for electrical fires or fires involving burning oils or flammable liquids. The extinguishing ability is practically the same as that of an equal quantity of water. The carbon dioxide evolved with the liquid is not produced for such an effect. Soda-acid extinguishers have ceased to be approved for firefighting underground as well as on surface due to their wetting characterstics.

Foam extinguishers

This type of extinguisher produces, when operated, a thick foamy fluid either by reaction between two chemical solutions (chemical-foam type) or by powerfully stirring an aqueous solution of a foam-forming compound by means of compressed air (mechanical or air-foam type). When directed against a fire, the foam spreads over the entire fire, covering or blanketing it, and thus cutting off the oxygen supply. The wet foam layer also produces a cooling effect which assists in extinguishing the fire. Foam is an extremely bad conductor of heat. It is to this fact that it owes its value. The topmost layer of foam has a temperature which is little above that of the surrounding atmosphere. All foams are capable of flowing and self-levelling over horizontal liquid or solid surfaces and adhering to vertical surfaces.

Foam extinguishers are especially used for fighting fires involving combustible and flammable liquids. They are not recommended for use in fighting electrical fires where the foam could come into contact with energized equipment.

Chemical foam extinguishers

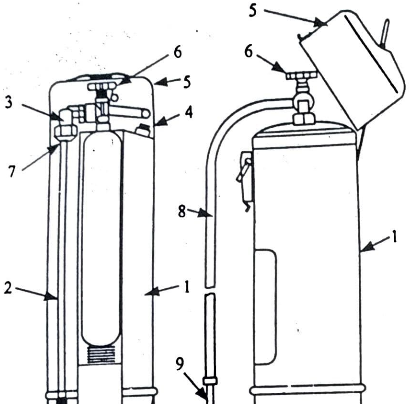
A chemical foam extinguisher consists of a cylindrical outer container filled with an aqueous solution of sodium bicarbonate containing a special foam-forming compound and a stabilizer, a cylindrical inner container containing an acid solution of aluminium sulphate, and a nozzle fitted to the outer container.

The extinguisher is operated by inverting it, permitting the acid and basic solutions to mix, and directing the nozzle, which may be fitted with a short length of hose, on to the fire to be extinguished. The chemical reaction produces gelatinous aluminium hydroxide and carbon dioxide according to the equation

$$6NaHCO_3 + Al_2(SO_4)_3 \longrightarrow 3Na_2SO_4 + 2Al(OH)_3 + 6CO_2$$

The carbon dioxide is liberated in a state of very fine bubbles which are maintained in that state by the foam-forming compound. The light gelatinous precipitate has the effect of making the bubbles more tenacious and permanent, thereby giving a very stable foam. The excess carbon dioxide under pressure serves to force the foam in a stream out of the nozzle. For the acid and basic solutions to thoroughly mix and form foam in the shortest time, a mixing chamber is provided before the nozzle.

Chemical foam extinguishers have been increasingly displaced by mechanical foam extinguishers in mines.

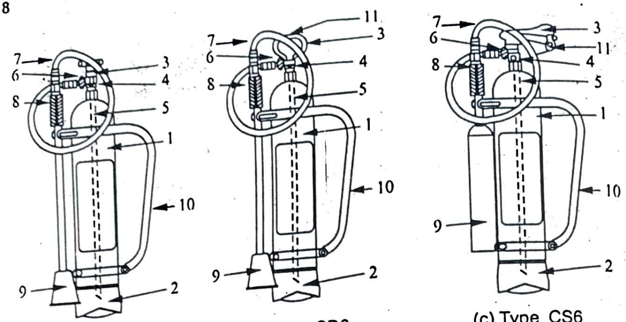


A 10-litre air-foam extinguisher produces with one filling about 120 to 140 litres air foam and has a discharge range of about 5 m. Two other sizes, 15 and 50 litres, are also manufactured for use in mines. Instead of compressed air, compressed carbon dioxide gas may also be used, but a jet pipe, a specialized discharge appliance, must be used which will suck the required quantity of air for foam formation and discharge it.

For continuous production of air foam, portable air-foam equipment, foam inductors, can be used in which the foam concentrate is continuously inducted into a water stream by means of an admixer or in-line proportioner located between a source of pressure water and an air-foam jet pipe or a foam-making branch. Figure 1. 14 shows the Minimax portable air-foam equipment.

The foam compounds now in general use with mechanical foam apparatus are of the hydrolyzed protein-type manufactured from horn and hoof meal or animal blood.

Carbon dioxide snow extinguishers


7
6
8
3
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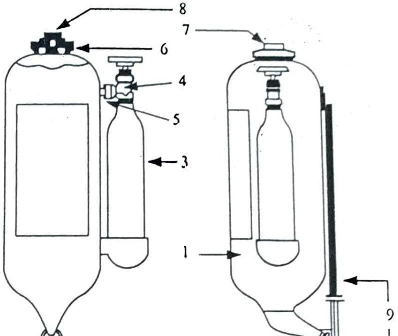
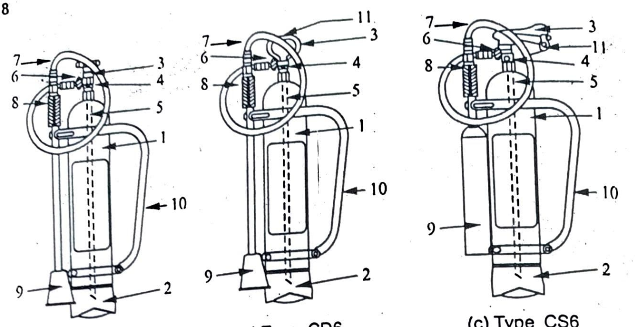
(a) Type C6 (b) Type CD6 (c) Type CS6

Minimax carbon dioxide snow extinguisher: 1: CO_2 cylinder; 2:
rupture disc and indicator; 5:

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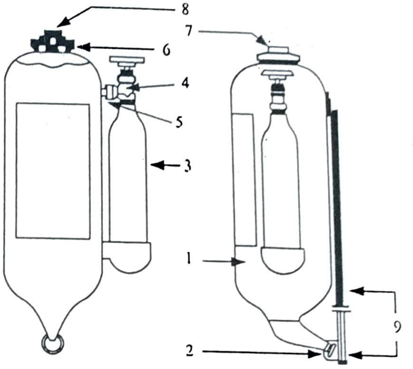
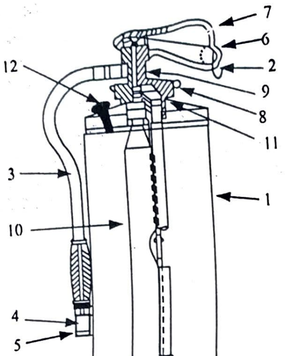
(a) Type C6 (b) Type CD6 (c) Type CS6

Figure 1.15 Minimax carbon dioxide snow extinguisher: 1: CO_2 cylinder; 2: standing base; 3: cylinder value; 4: safety device with rupture disc and indicator; 5: 7: high-pressure house; 8: hand grip; 9: discharge horn; 10: carrying





fires, it should not be assumed that they are valueless when used on other classes of fires.

For fires in substations, rectifier stations, battery-charging stations, pump chambers, and underground shops with electric-driven machinery, carbon dioxide snow extinguishers are the best. But dry chemical and multipurpose dry chemical extingushers can also be used although they leave a residue.

For fires in oil-storage areas, belt conveyor roadways and diesel mining vehicles, air-foam, dry chemical, and multipurpose dry chemical extinguishers may be used. Carbon dioxide snow extinguishers may also be used.

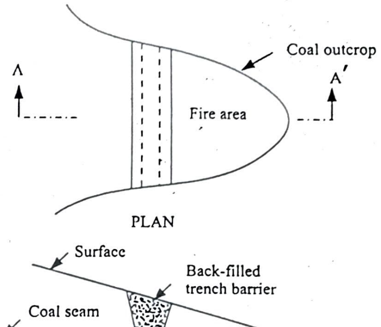
For smouldering fires and fires in ordinary combustible materials, air-foam and multipurpose dry chemical extinguishers may be used. For fires on self-propelled electrical machinery, dry chemical and multipurpose dry chemical extinguishers are the best.

For fixed automatic fire extinguisher systems which are necessary where fire hazards require immediate response, dry-chemical and multipurpose dry chemical currents so that the fire does not spread further but gets extinguished due to lack of oxygen.

1.1.4.2.1 *Isolation of fire*

Isolation of a fire by preventing air from gaining access to it depends on the location of the fire, the rate and direction of advance of the fire, and the nature of the combustible material. Isolation of a fire is easy in mines which are well-sectionalized, that is, divided into suitable fire sections by means of steel fire doors and preparatory stoppings at strategic locations. Each of the fire sections may be further subdivided into smaller sections. In laying out fire sections on the mine plans and sections, advantage should be taken of the presence of want or barren areas and large dykes.

Steel fire doors offer a safe, quick and easy means of isolating fires. They must be designed so that the door frames are as large as possible to reduce air resistance to a minimum when the doors are open and form an airtight seal.



of ventilation, the likelihood of formation of flammable gas mixtures, the coal dust hazard, the gradients in the area, the difficulty of transporting the materials of construction, balancing of pressures after sealing, and access to communication facilities. In general, the stoppings should be erected in sound (firm and unbroken) ground free of geological disturbances such as dykes, faults and burnt coal, as near to the fireseat as pratical without, however, sacrificing mine safety. Alternative plans for resiting the stoppings should also be prepared, should this be found necessary.

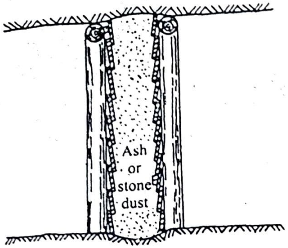
The type and construction of stoppings are governed by the condition of the ground such as friable adjacent strata, size of opening, rock pressure, availability of construction materials, danger of explosion, occurrence of acid mine water, and necessity to hold back water.

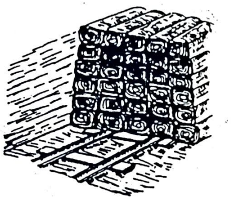
The sites for the stoppings should be prepared by digging hitches in the sides, roof, and floor (at least 0.5 m in rock and 1 m in coal), so that the stoppings are

the intake and return sides of the fire followed by permanent or main stoppings at selected palces for final sealing off. Experience had always shown that it is convenient to erect temporary stoppings first on the intake side instead of on both intake and return sides of the fire simultaneously as respiratory protective equipment will be necessary when men work in the return air. The temporary stoppings are erected with any available materials normally used within the mine that will require minimum time for construction with reasonable airtightness such as brattice line, sand bags, stonedust bags, boards, clay, light-gauge sheet metal and concrete or fly-ash blocks. It is advisable to seal off as small an area as possible so that the fire will reduce the oxygen quickly.

1.1.4.2.2.1.1 Temporary stoppings ✓

Temporary stoppings may simply be constructed of brattice line (jute, jute-plastic, nylon-plastic, propylene) if they are to be erected quickly and air leakage





roadway cross-section and boards placed between them and the sand bags. Sandbag stoppings, however, suffer from the following disadvantages: the sand bags cannot be stored as such underground; they may be burnt out by the fire, resulting in the running-off of the sand; they shrink in course of time causing leakage of air along the roof; and a large number of bags will be required.

Concrete and fly-ash stoppings

These can be built in a short time. A single or two courses of blocks are laid without mortar or with a weak mortar. The stopping is not hitched into the sides, roof, or floor. Its fireside is covered with a cement-sand plaster and the perimeter is caulked with cement mortar

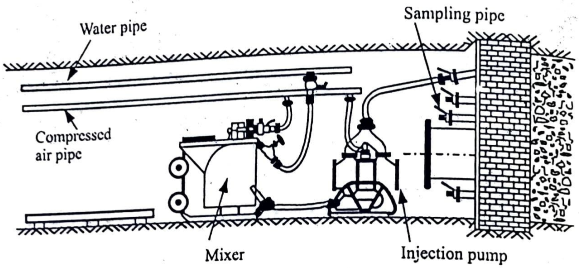
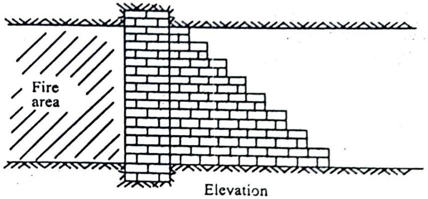
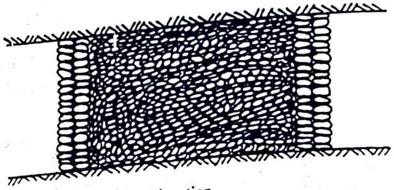

Water pipe
Sampling pipe
Compressed air pipe
Mixer
Injection pump

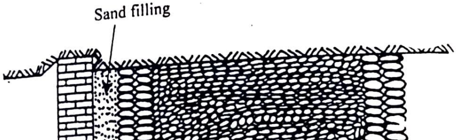
Figure 1.25 Slury injection pump



Elevation



Elevation



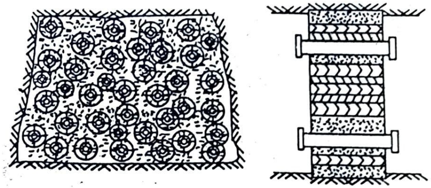
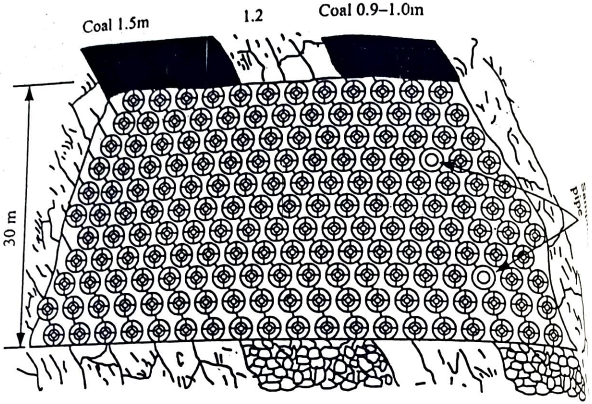


Figure 1.31 Post-and-loam single packwall stopping.

A layer of loam is next placed on the posts and is well tamped between them. Another layer of posts is added. The operations of putting alternate layers of posts and loam are repeated till the stopping is completed.

A double stopping (Fig. 1.32) is erected by constructing two single stoppings



difficult to interpret the analyses of air samples taken from behind the stoppings as the problems of balancing the ventilation pressures at stoppings increase and it is more difficult to control the air leakages.

(3) The area to be sealed should be cleared of pipes, cables, and other metallic materials.

(4) Samples from the fire area should be collected at shorter intervals of time and analyzed as quickly as possible in order that the stopping builders may be withdrawn, should the methane content near the fire become dangerously high. The result of the analyses should preferably be represented in the form a graph so that the make of gas can be seen at a glance during sealing.

(5) Under no circumstances must the ventilation be interrupted when building stoppings. Any changes in ventilation especially restriction or reduction of air quantities should be done with extreme care. It should be avoided that due to inadequate ventilation, firedamp accumulates or other dangers arise from migration of firedamp or firegases. To maintain sufficient ventilation through the fire area until the time for sealing off and also to enable rescue crew to pass through for purposes of

(10) The pressure at stoppings must be balanced after sealing.

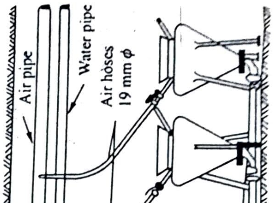
(11) All events should be logged under the general heading 'Time, Event, Action Taken' and the records kept in case the sealed area is to be reopened at a later date.

(12) Telephonic communication between all stoppings, the fresh-air base, and the surface control room is desirable.

(13) Alternative plans for resiting of stoppings should be prepared if found necessary.

1.1.4.2.2.2.1 *Temporary stoppings* ✓

As temporary stoppings, sand-bag or stone dust-bag stoppings of 4 to 6 m thickness have been found to be effective.



with a total labour force of 8 to 10 men. About 16 to 20 manshifts are necessary for the erection of a stopping which may take 25 to 35 t of gypsum.

According to British practice, 75 mm × 75 mm wooden frames are erected across the roadway at about 1m intervals. The inbye frame is covered with fire-retardant slabs which are shaped to allow pipes to pass through the stopping. The following frame is covered in the same manner, working upwards to the roof as required, allowing filling of a section by pumping slurry between the two shutterings. After completion of the stopping, the surrounding strata are injected with gypsum (Hardstop). The thickness of the stopping is taken as equal to

$$\frac{\text{Perimeter of the roadway}}{2} + 0.3 \quad \text{(m)}$$

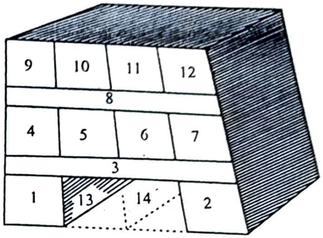


Figure 1.35 Wooden wedge stopping.

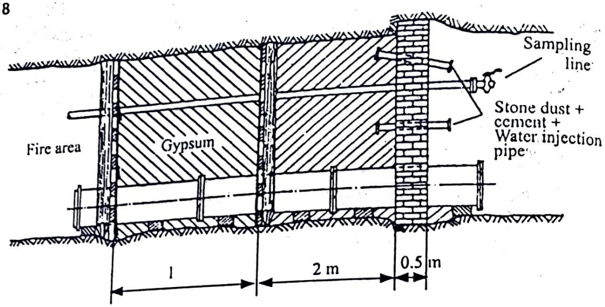


Figure 1.37 Gypsum stopping.

The results of each inspection should be noted on a board hung near each stopping and recorded in a register kept for the purpose. Samples of air should also be collected through sample pipes of one or more suitably located stoppings to determine the status of the fire. Any unusual circumstances should be immediately reported to the mine manager.

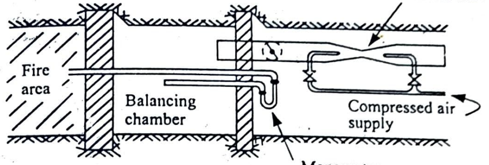
In detecting air leakages at stoppings, one has to deal with very small air velocities which are difficult to determine.

When testing for air leakages, one or more of the following methods are frequently used:

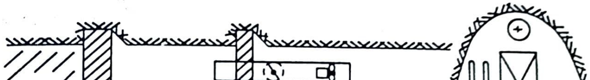
(1) Feeling the stopping to find out if there are any hot or warm spots; hot or warm spots may not necessarily indicate the temperatures within the sealed area but they are definitely a pointer to a high temperature condition in the area;

(2) Detecting any hissing sound by putting an ear to the stopping;

stoppings. When pressure on all sides of a sealed-off area is not the same due to the peculiar location of the area with respect to the ventilation system or pressure loss in airways, uncontrolled leakage of fresh air from the intake into the area and leakage of methane and noxious firegases out of the area may take place through leaky stoppings as well as through cracks and fissures in the strata around the stoppings. Even if the pressures on the stoppings are balanced by removing or resiting of doors, erection of further stoppings, air-crossings, regulators, pressure-balancing roadways or ducts, or fans (booster and auxiliary), leakages through them may take place when the pressure in the sealed-off area is less or greater than the pressure outside the stoppings. Under such conditions, it will be necessary to set off the area from the rest of the mine and provide a balancing chamber at each leaky stopping to enable the pressure on the outbye face of the stopping to be controlled for a long period of time. After sealing off, advantageous to conduct a pressure survey local to the stoppings



(a)



(9) When flooding the entire mine, existence of adequate barriers between it and the adjoining mines to withstand the head of water in the flooded mine; and
(10) Loss of entire mine production.

1.1.4.2.5 *Hydraulic flushing of enclosed area with sand and other materials*
The method can be recommended where the strata are so broken that neither sealing nor flooding will be effective. Generally, the fire is sealed off with stoppings in the usual manner and then the sealed area is flushed with a clay-water, sand-water, sand-crushed rock-water, or sand-clay-water-mixture through pipes left in the stoppings. In metal mines, mill tailings as a sludge may be used. Experience has shown that solids fill the voids completely, reduce and, in certain places, eliminate fire gases, cool the adjacent rocks, and offer support to the ground.

During flushing, care should be taken so that the slurry is not directed straight on to the fireseat but at a point near it so that water gas explosion from dissociation

(4) The location of stoppings is of secondary importance in fighting a fire if the inert gas could be injected early. An inerted fire area is recovered in stages by advancing the intake stoppings with continuous flushing out of the gases from the fire area with the inert gas. When this is done, stoppings on the return side may be eliminated.
(5) The time interval between sealing and recovery of a sealed-off fire area is shorter due to faster cooling off of the area. It may be possible to effect a smooth transition between sealing and reopening of a fire area.
(6) Recovery work is negligible and, hence, rapid.
(7) It is possible to work without the use of breathing apparatus.
(8) The rescue and recovery crew work under improved working conditions with reduced temperatures and relative humidity.

For inertization of mine fires, the following practices are adopted:

(1) Inertization by flushing the fireseat with inert gas maintaining normal ventilation in the working district/section (Local Inertization); and
(2) Inertization by flooding a sealed-off area with inert gas. (Area or

With area or zone inertization (Fig. 1.41), the inert gas is introduced into a fire area which has been sealed on all sides or at least on the intake side with the object of rendering flammable gas mixtures non-flammable, containing the development of the fire, and recovering the mine workings expeditiously. It is mainly used against open fires and has been employed in only a few individual cases of incipient fires to eliminate the danger of an explosion in the goaf. To fight open fires, the oxygen content of the air reaching the fireseat must be reduced to below 10 per cent by volume. The inert gas requirement is usually between 100 and 500 m³ N_2/min.

Under favourable conditions, a fireseat in the goaf can be effectively fought by local inertization following area inertization, which will eliminate the risk of explosion. A saving in the construction of cost fire-stoppings may be achieved.

1.1.4.2.6.1 Inert gases for inertization

The inert gases which have been successfully used for inertization are carbon and nitrogen

Combustion gases

Combustion gas produced in a combustion-type inert gas generator burning fuel oil as in Poland, or in a kerosene-fuelled jet engine, as in the ertswhile USSR, Poland and Czechoslovakia, may be used to provide an unlimited supply of inert gas. The generators are operated above ground and the inert gas is forced through a pipeline by a compressor, whereas the jet engines are located underground in an airway through which the entire intake air current to the fire must pass. The combustion gas consists of about 80 per cent N_2, 13 per cent CO_2, and the remaining residual oxygen; CO, SO_2, NO_x, and water vapour. In Eastern Europe, inert gas generators and turbines with capacities between 100 and 1500 ml/min have been used.

The disadvantages of the use of combustion gases for inertization of mine fires are:

(1) The gases have a high temperature requiring large quantities of water for cooling

(5) It is slightly cheaper than carbon dioxide and more costly than combustion gases.

For direct application of liquid nitrogen for fire control, the nitrogen is transported to the place of use in insulated containers and then atomized behind the fire-stopping using nitrogen-atomizing nozzles. The advantages claimed for this method are that the positive pressure resulting from vaporization prevents air leakages into the fire area and that the heat of vaporization is drawn from the fire gases and the adjacent strata, thereby exerting a cooling effect. In shallow mines, liquid nitrogen may be introduced into the fire area through a surface borehole to reduce the risk of explosion while building fire-stoppings.

Inertization with gaseous nitrogen obtained by vaporising liquid nitrogen at the mine site is the most commonly used method of combating mine fires.

Production of nitrogen

Liquid nitrogen in bulk is produced at major chemical works by the Linde air liquefaction process. Recent developments centre round production of small- to medium-size plants based on special techniques of nitrogen production.

The pressure swing adsorption plant operates on the principle that compressed ambient air separates into nitrogen and oxygen-rich gas when passed through whose internal pore system N_2 and O_2 diffuse

(2) Ambient-air evaporators.

Water-bath evaporators operate on the principle that liquid nitrogen vaporizes as it passes through pipe coils in a heated water bath. The optimal temperature of a water bath is 20° to 30°C. About 600 kJ are required to vaporize 1 kg of liquid nitrogen and warm it up to +5°C. They have capacities of 65, 100, 125, and 300 m³ N_2/min. They can be regulated to dispense nitrogen flow rates between 5 and 120 m³ /min.

Air evaporators consist of nests of finned tubes of aluminium through which liquid nitrogen passes and vaporizes by drawing heat from the ambient air. After-heaters are used on the downstream side to warm up the gas 0° to 15°C. Electrical trim heaters with or without water baths are used for this purpose.

Transmission of nitrogen underground

When contemplating fighting mine fires with nitrogen, an 'inertization plan' must be prepared which covers not only active working faces/districts but also drivages using fast-heading machines. Besides setting out actions to be taken on the surface and underground, the inertization plan clearly specifies which of the existing pipe-ranges in shafts and pipelines in roadways can be utilized as inert gas lines in an emergency without affecting the firefighting operations. The position of the inert

1.1.4.2.6.2 *Monitoring the results of inertization*

In order to monitor the results of inertization and to determine the success of firefighting, the composition of the gases in the fire area and the ventilating air currents immediately downstream of the fire area must be continously monitored.

For this purpose, continuous remote environmental monitoring and control systems will prove to be most effective. This may be supplemented by collecting gas samples at regular intervals and analyzing them immediately. A sufficient number of portable gas-measuring/monitoring instruments such as detector tubes and O_2–, CH_4–, and CO– hand-measuring instruments should be made available to those engaged in firefighting and/ or recovery operations. Also, the volumetric flow-rate and pressure and temperature of the nitrogen gas must be monitored before its entry into the shaft range.

Experience with inertization of mine fires has shown that the success of the method depends on the speed with which inertization can be applied. A maximum of 12 hours for all the preparatory jobs may be allowed. It must be borne in mind that the inertization method will not displace the conventional firefighting methods. A deep-seated fire will almost never be completely extinguished by

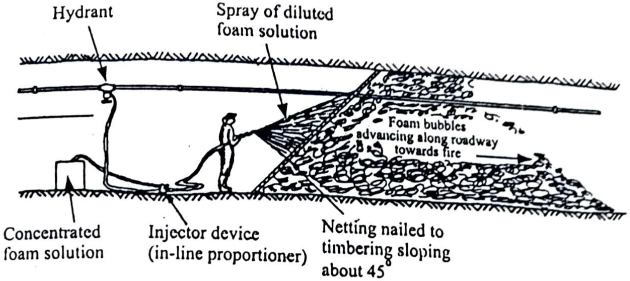

Hydrant
Spray of diluted foam solution
Foam bubbles advancing along roadway towards fire
Concentrated foam solution
Injector device (in-line proportioner)
Netting nailed to timbering sloping about 45°

Figure 1.42 High-expansion foam plug (SMRE).

on the intake side (Fig. 1.42). The ventilating current passing through the net which is continuously wetted, produces on the return side of the net bubbles which coalesce to form a mass of foam that quickly fills the roadway, creating a

(7) The solution must contain at least 2 per cent (by volume) foaming compound. The rate of feeding the solution must lie between 0.002 and 0.001 of the volumetric rate of air supply.

To increase the scope of application of the high-expansion foam in fighting fires, the US Bureau of Mines perfected a practical foam generator that forces foam directly to the fire through a plastic tubing independent of mine ventilation for making the foam-plug. Mine Safety Appliances Co., the USA brought out transportable foam generators or foamakers into the market that will fit into the category of secondary firefighting equipment to fight localized mine fires not accessible to direct attack with conventional firefighting equipment and might have to be sealed off. The foam can be produced at such a rate that the time of travel between the foam generator and the farthest limit of a fire is three hours.

The MSA Mine Model Foamaker (Fig. 1.43) is available in two models with foam output capacities of about 85 m³ (3 000 cfm) and about 113 m³ (4000 cfm) as rubber-tyre or skid-mounted units. It consists of a 3.5-hp axial-vane

is provided with a louvre at its intake end to control the inlet air supply and, hence, the rate of foam production, while at its discharge end a screen of polyester fibre cloth or fibre-glass netting is fitted which is sprayed with a mixture of foam concentrate and water using full cone nozzles. The fan aerates the mixture into a frothy and bubbling foam which is then driven through the tubing. A flow of 136 litres per minute of water at an inlet pressure of 2 bar will be required. A foam proportioning pump adjusts the quantity of the foam concentrate, depending on the quantity of water used and the length of tubing required.

The advantages of transportable foam generators are:

(1) They can be quickly put into service within a minute of their installation;
(2) Being simple and rugged in construction, they can be easily operated by non-technical personnel;
(3) The full water content of the foam reaches the fire and is not drained off during transportation; and

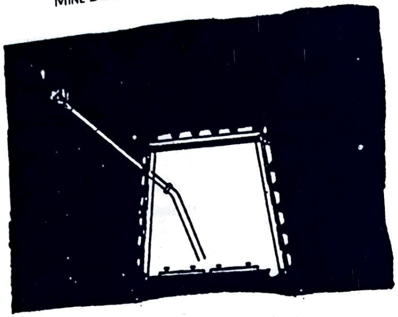


Figure 1.44 Fighting seam fire with antipyrogene injection.

air or gas cylinder

The reversal of mine ventilation following a mine fire is a controversial subj The important factors to be considered before reversing the direction of air flow are:

(1) Stage of fire;
(2) Location of the fire with respect to the ventilation system;
(3) Presence of workers in mine workings;
(4) Type of fan; and
(5) Type of doors.

When a fire in a fiery and dusty mine assumes serious proportions and threatens to engulf the entire mine in a series of mine explosions, one should be prepared for all eventualities.

As a general rule, the main fan should not be reversed unless the fire is in the junction of the main intake splits, in the downcast shaft, or near the pitbottom of a downcast shaft. With fire in a timbered downcast shaft, reversal of ventilation impossible to establish and nothing is known of air directions. The major decision on stoppage or slowing down or reversal should be taken only after thorough consulations with all the senior mine officials involved.

It should be remembered that withdrawal or evacuation of men should be the foremost concern of the mine management before any one of the above methods of controlling ventilation is adopted and also that the method of ventilation chosen may be modified during firefighting if conditions so warrant.

1.1.4.2.9 *Firefighting organization*

As fire hazard is always present, every mine, no matter how small, should perfect and maintain its own firefighting and rescue organization to ensure maximum safety and escape strategy by:

(a) Training rescue crew and other personnel through a continuing education programme;

fire. Training of a few selected personnel in proper use of all available firefighting facilities is essential in organizing effective firefighting services at mines. The need for training with use of gas masks and breathing apparatus is obvious. Training will create a consciousness of fire and fire hazards among all mine personnel. A record of the training programmes must be maintained by the mine management.

Procedure after fire

At the appearance of any sign of fire (sweating, petrolic smell, tarry smell), the workman who notices it should inform the nearest supervisory official as soon as practicable. With the discovery of an open fire, he and any other persons in the vicinity of the fire should make a prompt effort to extinguish it with water, stone dust, or hand fire extinguisher and inform the nearest supervisory official without delay. The official who is warned of the occurrence of a fire should immediately inform the manager of the mine or, in his absence, the principal official in-charge.

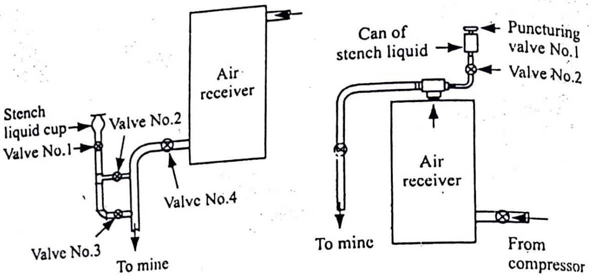

Air receiver
Stench liquid cup
Valve No.2
Valve No.1
Valve No.4
Valve No.3
To mine
Can of stench liquid
Puncturing valve No.1
Valve No.2
Air receiver
To mine
From compressor

Figure 1.46 Stench injector devices

its travel over long distances in steel pipes

be equipped with either a stairway (ladderway) or cleats, and when the inclination is more than 45°, only a ladderway should be installed. An escapeway plan should be hung at a suitable place in each working section. The withdrawal of workmen should be accomplished in an orderly manner so that every man may be accurately accounted for. Computers may be used to simulate evacuation of a mine in a variety of emergency situations without conducting fire-drills. It is possible to predict various mine evacuation times for the different evacuation strategies.

The manager or, in his absence, the principal official present at the surface, should immediately set the mine rescue alarm plan into action and set up emergency underground and surface organizations as outlined in Chapter 7.

Firefighting facilities

The efficiency of a mine firefighting organization depends on the availability of adequate and proper firefighting facilities at suitable places underground as well as on their constant maintenance so that they with his experienced colleagues who are conversant with local conditions and are well-versed in the procedures following mine fires. The plan provides for the removal of endangered workmen if they have not been removed or warned earlier, pinpointing the location of the fire and the strategy of firefighting, selection of suitable sites and type of stoppings and their order of erection. Some exploration work by mine rescue or gas mask reconnaissance crew may be necessary to throw some light on the situation. The plan must be strictly followed with no essential deviation from it. The manager should direct the firefighting operations from a control room on the surface. If the manager is of the opinion that his presence underground is necessary at times, he should keep the control room, in his absence, under the charge of a responsible senior mine official.

[illegible] procedures must be kept under constant review and innovations

The possible causes of ignition are:

(1) Spontaneous combustion; this is the most common cause of ignition of refuse banks and coal stacks as well as of coal faces in opencast mines;
(2) Careless burning of trash and/or garbage from nearby dwellings;
(3) Fires due to burning of dried vegetation when not insulated from the refuse pile;
(4) Underground fires in shallow mines reaching the surface through cracks due to subsidence;
(5). Dumping of hot ashes;
(6) Intentional ignition to create blay which may be used as road base materials; and
(7) Camp fires left burning; and illicit distillation of liquor.

(8) Inspecting the pile regularly to detect fumes, the characteristic smell of gases, etc. from hot spots.

1.2.2.2 Fighting coal refuse fires

Various methods or techniques are available to fight refuse fires. The extinguishment of a burning refuse fire is a difficult and costly operation. The selection of a method or a combination of methods depends on the size and nature of the pile, progress of the fire, the availability of suitable extinguishing materials and equipment, and proximity of homes. Each fire represents an individual problem and one must tailor a technique selected to suit the given situation.

Experience has shown that the following methods can be used:

Water applied internally into the pile through pipes inserted into it has limited success in extinguishing the fire as, after treatment there is usually recurrence of fire on a larger scale, probably due to the creation of additional passages for air to enter by the water draining away.

Blanketing

It aims at cutting off air supply to the burning area by applying a blanket or covering seal of an incombustible material such as limestone dust, fly-ash, shale dust, earth, or clay. It is often effective if the blanket is sufficiently thick (greater than 3 m), completely covers the affected area (particularly the lower parts of the surrounding slopes), consolidated, and is maintained airtight.Sand alone does not form a good blanket unless covered with a clay layer. Constant inspection

(1) A programme of gas testing by a gas detector. Action to be taken when and if gas is found and rescue of workmen decided upon.
(2) Suitable breathing apparatus should be available for all those immediately connected with the work in the danger area.
(3) Machine operators should be instructed on the action to be taken in the event of a machine breakdown in the danger area.
(4) A telephone or other means of communication should be provided.
(5) A first-aid worker should be in attendance.
(6) A hand-held IR-instrument should be provided for locating hot spots and temperature gradients in the pile. It will be extremely useful in developing a proper firefighting strategy.
(7) At the end of a day, all men should be accounted for.

Height of stockpile
The height of a coal stack depends on the strength of the coal, liability of the coal to spontaneous combustion, and whether it is deposited in layers with compaction or not. Uncompacted piles should not normally be stacked to heights greater than 3 m; with compaction, the height can be raised to 10 m or more, provided proper supervision and scientific control are exercised.

Selection and preparation of site
The site of a coal stack must be carefully selected, taking into account the quantity of coal to be stored, cost of preparation of the site, methods of storage and reclamation, and drainage. It should be located away from any source of heat such as steam or hot water lines, furnaces, smoke stacks, or any refuse bank which has a history of heating.

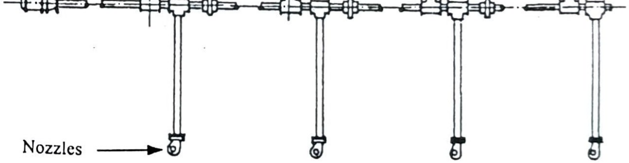
The site should be free of all brush, growth or trash, mounds and ditches.

To check the temperature in the pile at reg months of the life of the stacks, 25-mm capped steel pipes closed at the botto may be driven on 3 to 6 m centres down to within 30 to 60 cm of the bottom o the pile. An armoured-type of maximum reading thermometer may be used t measure the temperature at varying depths. Alternatively, a series of thermocoupl may be inserted into the pipes. A practical method of detection that may be use consists of introducing long steel rods in suspected hot places in the coal, leavin them for a few minutes, withdrawing them, and feeling their temperature.

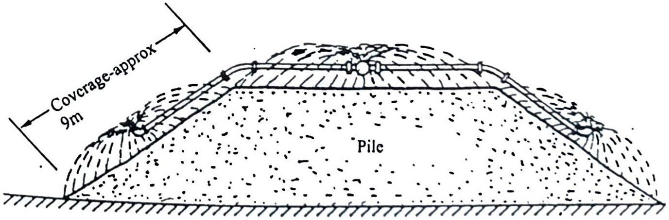
To eliminate chimney effects created by vertical pipes, horizontal pipes m be driven to monitor the temperatures, which may require reclamation from t downwards.

When the coal temperatures are less than 35°C, temperature readings may b taken fortnightly but when the temperature exceeds 35°C, they may be take weekly. When the temperature reaches 45°C, consultations should be held t chart out a future course of action to deal with the situation as spontaneou combustion is liable to occur as the critical temperature of about 70°C is reache

Thermal infrared imaging technique may be used to identify higher th ambient temperatures in the pile.



Top view (not to scale)



Side view

Figure 1.48 Water-spraying a pile

spray nozzles will suffice for this purpose.

1.2.5 Fires in opencast coal mines

Fires occur during mining of thick bituminous and non-coking coal seams liable to spontaneous heating and brown coal seams by the opencast mining method. The conditions favourable for occurrence of fires in hard coal mines are:

(1) presence of micro- and macro-cracks in massive coal benches providing greater access of atmospheric oxygen to coal surfaces;
(2) long exposure of coal benches to the atmosphere; and
(3) accumulation of loose coal at the toe of the benches.
(4) availability of pyrite-bearing coal seams, reactions with ANFO explosive charges with weathering-oxidation products of pyrite.

producing intense heat sufficient to cause an outbreak of fire in combustible materials in contact with or in the immediate neighbourhood, producing significant quantities of toxic sulphur dioxide and carbon monoxide. Owing to the generation of a large amount of heat, extremely difficult environmental conditions may be created at the workplace and, when associated with crushed decaying timber, the fire may assume serious proportions which may be difficult to control.

The sulphide oxidation process is complex and depends largely on physical, chemical, and electrochemical factors which interdependently combine in the oxidation process. The important factors are the ore fineness, moisture content, water content of ore, porosity and permeability of the ore, temperature, depth of mine, ventilation, acidity of mine water, oxygen availability, minerological composition, sulphide type and sulphide content, and association with different sulphides and gangue minerals.

Fires due to spontaneous combustion in sulphide ore mines occur in mined-out stops containing highly fractured backfill with a large sulphide content with in Canadian and Australian mines. Sulphide-dust explosions have occurred in development blasting in a Canadian base metal mine belonging to Brunswick Mining & Smelting Corporation Ltd. Accidental explosions in sulphide ore mines using ANFO explosives have been ascribed to the reactions between ANFO explosive charges and pyrite oxidation products.

In India, between 1900 and 1980, 21 explosions occurred, the worst disaster being that in Dhori Colliery (1965) where 268 miners were killed.

2.1 Firedamp Explosions

Methane present in coal mines is a by-product of the coalification process during which the coal was formed from vegetable matter. It exists in the coal seam both in an adsorbed state, adhering to the internal micropore surface of the coal matrix, and in a compressed state in the fracture system of the seam. It is also present in the adjacent strata compressed in the fracture system of the rock. When the seam is mined, the equilibrium that existed in the coal seam and the surrounding strata

2.1.1 Limits of flammability, flammable limits, or explosive limits

The flammable limits of methane–air mixtures are the limits of concentration of methane in air between which a flame can be propagated throughout the mixtures. The boundary-line mixtures with minimum and maximum concentration of methane in air, which if ignited, will just propagate flame are known as the lower and upper flammable or explosive limits. Flamiability limits of methane have been extensively studied. The lower and upper flammable limits (LFL and UFL) are approximately 5 (33 g/m³) and 15 (100 g/m³) respectively. Explosions at the flammable limits are rather weak and slow-burning, while at the stoichiometric concentration, they are most violently explosive. A mixture containing less than 5 per cent methane, although not flammable under normal conditions, may explode when at a high temperature or when compressed adiabatically due to blasting or a coal-dust explosion. Also, a mixture containing 15 to 100 per cent methane in which electric sparks can be generated without any explosion taking place is dangerous in a mine as it may pass through the flammable range on dilution with air in mine workings.

The limits of flammability are not fundamental characteristics of the gas but depend on experimental conditions. They are influenced by the presence of hydrogen, etc., which have, like methane, lower and upper flammable limits also reduces the lower limit which can be determined by using the Le Chatelier relation:

$$\boxed{\frac{100}{L} = \frac{p_1}{l_1} + \frac{p_2}{l_2} + \frac{p_3}{l_3} + \cdots}$$

where $p_1, p_2, p_3, \ldots$ are the percentages of the component gases in the mixture ($p_1 + p_2 + p_3 + \cdots = 100$ per cent) and $l_1, l_2, l_3 \ldots$ their percentage lower limits.

The presence of inert gases has a damping effect on the flammability of methane–air mixtures. Carbon dioxide is more effective than nitrogen.

2.1.2 Ignition point or ignition temperature

The ignition point of flammable firedamp–air mixtures is given as 650° to 750°C. The ignition point is the minimum temperature to which a portion of the mixture must be raised in order to initiate or cause a rapidly accelerating reaction in the whole of the accumulation with the accompaniment of a flame. It does not refer to the temperature of the igniting source which must obviously be at a higher temperature. It is not a definite temperature but depends upon the nature of the source of ignition whether flame, spark, etc and shape and size of the space

Explosion pressure

The explosion pressure depends on the flame temperature and confinement. The maximum explosion pressure of a methane–air mixture (760 mm Hg, 20°C) when ignited in a closed vessel is given as 7.2 bar(g) [26]. In mine workings, however, the explosion pressure may exceed this value. The shock wave (pressure wave) travelling ahead of the explosion flame will compress any firedamp accumulation encountered on its way to a pressure greater than 1 bar(g). Upon ignition of such an accumulation, the maximum explosion pressure of the resulting explosion will be much greater than 7 bar(g). This explains the observation generally made that severe damage from a firedamp explosion does not occur at the seat of the explosion but away from it.

Experiments on gas explosion phenomena in the US Bureau of Mines (USBM) Experimental Coal Mine [27] using natural gas (83 per cent CH_4, 16 per cent C_2H_6, and 1 per cent N_2) showed that the highest explosion pressure occurs in the gas zone and that the pressure decays exponentially with distance outbye the gas zone. Flammability data for quiescent methane–air mixtures in a 20–*l* flammability test chamber using a strong electric spark as the ignition source obtained by the US Bureau of Mines [28] are shown in Fig. 2.2. The peak explosion pressure ratio is the peak explosion pressure divided by the initial pressure of the mixture.

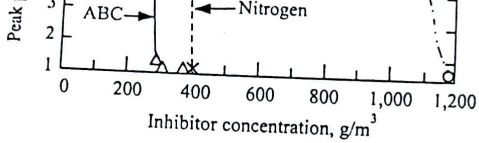


Figure 2.3 Effect of inhibitors on explosion pressure of stoichiometric methane–air mixture in a 8–*l* Test Chamber.

Figure 2.3 gives the flammability diagram for stoichiometric methane–air mixtures with added inhibitor nitrogen gas and dusts in an 8–*l* flammability test chamber, in which the order or relative effectiveness of the various inhibitors in extinguishing methane explosions is obtained as $NH_4H_2PO_4 > KHCO_3 > Al_2O_3 = CaCO_3$. Table 2.1 gives the inhibitor concentrations to inert a stoichiometric methane–air mixture [29].

(2) *Condition of the gas mixture,* whether it is found from experiments conducted in quiescent mixtures in tubes that the maximum velocity of propagation is not greater than 0.6 m/s [25]. On the other hand, when the gas mixture is in motion, the velocity may quickly increase to a few hundreds of metres per second.

(3) *Point of ignition* The location of the point of ignition within the body of the gas mixture markedly affects the velocity of propagation. If ignition takes place at the closed end of a roadway filled with a gas mixture, the speed of propagation and severity of explosion are greater than when ignition takes place at the outbye end of the roadway. When ignited at the outbye end, rapid burning rather than an explosion develops. It is for this reason that the 'fire boss' in the early days of mining could intentionally ignite gas as he entered a heading without igniting a dust explosion.

(4) *Length of gas zone* Experiments in gas zones in the USBM Experimental Coal Mine [27] showed that the velocity of flame propagation of a gas explosion increases from zero at the point of ignition to a maximum at a distance of about twice that of the original length of the gas body; the

The hazard posed by firedamp to mine safety may be estimated by:

(1) determining the gas emission indices or rates in return air currents (m³ CH_4/t of r.o.m or saleable output) in coal mines;

(2) determining the concentration of methane at working faces; and

(3) the number, type and size of firedamp accumulations.

The gas emission index or emission rate is determined by using the formula

$$q_{CH_4} = \frac{c \times 60 \times 24 \times Q_{air}}{100 \times O \times K} = \frac{c \times 14.4 \times Q_{air}}{O \times K}$$

where c = methane content in return air current (Vol. %)

Q_{air} = return air current quantity (m³/min)

O = mined or saleable output (t/d)

K = a factor given by the ratio $\frac{\text{max.conc.of methane on any day in a week}}{\text{average concentration of methane in the week}}$.

III	10–15
Outside the categories	> 15

For gassy non-coal mines, any one of the coal mine classification criteria is usually applied or no classification criteria exist.

A firedamp accumulation in a mine working is said to take place when the percentage of firedamp in the general body of the air current flowing in the mine working is more than that prescribed by the mining regulation (1.25 per cent in India, 1 per cent in Europe).

2.1.5 Causes of firedamp explosions

The greatest number of firedamp explosions occur in active mine workings in face areas and headings. The various causes of firedamp explosions in mines may be grouped under the following headings:

(a) Negligence of miners
(b) Use of damaged safety lamps and their improper handling
(c) Blasting

frictional sparks can, under certain circumstances, ignite flammable firedamp–air mixtures. Frictional ignition in mines may take place due to: (a) friction between metal and metal, (b) friction between metal (especially steel) and rock, and (c) friction between rock and rock.

Frictional heatings are seldom the cause of ignition of flammable firedamp–air mixtures.

The degree of ignition hazard from metal-to-metal contact is determined by the properties of the more readily oxidizable metal. Extensive research carried out in the erstswhile FRG and the UK on the possible ignition of flammable firedamp mixtures by incendive metal sparks has clearly shown that the ignition hazard from the incendive particles of iron is extremely less and that danger may be feared from aluminium and its alloys [30].

Friction between metal and rock poses the greatest ignition hazard in coal mines using cutting machines (cutters, continuous miners, power loaders). Several firedamp explosions in mines in recent years have been attributed to frictional sparking during cutting and drilling operations in seams containing hard siliceous bands or inclusion of iron pyrite nodules. Investigations showed that the nature of the rock, particularly the quartz content and its grain size, is source of ignition – electric sparks igniting not only combustible also flammable firedamp–air mixtures. Sparks may be produced from switchgear, damaged cables, signalling apparatus, and faulty electrical equipment. Although electric sparks usually have a much higher temperature than ordinary flames, it may happen that a spark has a very short life and its electrical energy is not sufficient to cause ignition of the mixture in that time. The minimum energy of a spark causing ignition of a flammable firedamp mixture varies with methane concentration, humidity, oxygen content of the atmosphere, temperature, pressure and turbulence.

The variation of the minimum ignition energy with different methane–air mixtures is shown [32] in Fig. 2.4, from which it is seen that the most flammable mixture for ignition by an electric spark occurs at 8.3 per cent methane, irrespective of the type of spark (low-voltage inductive or high-voltage capacitive) and that the minimum ignition energy is as low as 0.28 mJ (mWs)

Other special causes

Some of the mine explosions in recent years have been ascribed to ignition of firedamp– or dust–air mixtures by sparks of static or frictional electricity having adequate energy which will be some fraction of the total stored energy. In gassy coal mines, electrostatic charges in excess of 300 V must be viewed with caution.

by friction.

Methane blowers, when they catch fire, can become a cause of mine explosions.

2.1.6 Prevention of firedamp explosions

No practical measures exist for arresting firedamp explosions in coal mines. The only thing that can be done is to step up protective measures.

The various measures against the firedamp explosion hazard in mines can be divided into the following three groups:

2.1.6.1 Measures against accumulation of dangerous firedamp mixtures in mine workings from the beginning

The most effective method of preventing firedamp explosions in mines is by providing adequate ventilation which will dilute the firedamp, besides other harmful gases, to well below limits that may be prescribed for different mine workings and carry it away to the surface. Frequent sampling of mine air for methane at several points in the mine is, therefore, an important measure for prevention of firedamp explosions.

men, equipment and trains are passing through them. They should always be in good repair and be self-closing.

(vi) The mine ventilation system should be planned so that simple, effective, and reliable ventilation of all workings is assured. Where multiple main fans are used, the ventilation system should be so arranged that no adverse air reversal will occur in the event of failure or stoppage of any fan or fans. Predictions of firedamp emission for different sites in a coalfield will greatly assist in planning safe ventilation systems for mines.

(vii) Seams should be extracted, as a rule, from top downwards to decrease the methane levels in the lower seams.

(viii) The method of extraction should be selected so that it guarantees an easy and safe ventilation of the faces by air dilution with adequate velocities at the face and at the waste edge. Face air velocities of up to 4 m/s or more are being used today on modern longwall faces in very gassy seams to mine coal safely.

In high-capacity longwall faces with strong methane emissions and high ventilating air requirements, faces laid with W-ventilation system will enable larger air quantities to flow through them besides providing a middle gate for drilling additional methane drainage holes.

(ix) A particularly high standard of unit ventilation by separate ventilation splits should be maintained in each mechanized mining section and in districts/panels liable to gas outbursts.

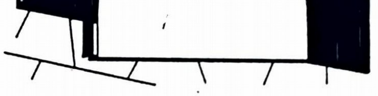


Figure 2.6 Roof voids/cavities filled with foamed plastic.

(x) In bord and pillar and longwall retreating panels in very gassy seams worked with caving, the firedamp content in the goaves behind the active faces must be controlled by in-mine local or central drainage of goaves or drainage through surface ventilation boreholes. Bleeder systems with bleeder entries and bleeder entry connections as in American mining practice can be successfully employed to move firedamp–air mixtures continuously from the caved areas behind retreating longwall faces away to the return airways, and to minimize the hazard of expansion of goaf gases due to changes in atmospheric pressure (Fig. 2.5) and their release into faces and gate roads as unusual emissions during the rapid closure of the goaf, just behind the face supports or not far away from them, when caving takes place periodically.

(xi) Mining with backfilling (Fig.2.6) or solid stowing of the waste/goaf, especially hydraulic filling, prevents formation of methane reservoirs in goaves.

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A correction factor must be used for the rising inclination and change in the cross-sectional area of the airway.

2.1.6.2 Measures against ignition of flammable firedamp mixtures

The various preventive measures to be taken against ignition of flammable firedamp mixtures in mines are:

(i) All persons should be prohibited from carrying smoking articles (pipes, cigars, cigarettes, tobacco other than chewing tobacco or snuff), matches or other spark- or flame-making devices into the workings. All men entering the mine should be searched for contraband.

(ii) All coal mines should be treated as safety-lamp mines as a number of explosions in the past had occurred in the so-called naked-light mines. In short-life naked-lamp mines where naked lights are to be retained, special attentions should be paid to ventilation, gas-testing, and to precautions against coal dust.

be immediately cut off in return airways. After the fan has been restarted, the power shall not be switched on unless normal ventilation and safe conditions have been restored.

(h) In places where auxiliary fans are used, the auxiliary fans should not be operated during stoppage of normal mine ventilation. Accumulations of methane should be removed after restoration of normal mine ventilation before the fans are operated. In a place ventilated with an auxiliary fan, if the auxiliary fan is stopped or fails, electrical equipment operating in the place should be stopped and the power disconnected at the power source until the ventilation is restored. The auxiliary fan should be maintained so that the impeller does not strike the casing.

(i) Changes in ventilation affecting the main air current or any split thereof should be made only when the mine is idle. The power supply should be cut off from the affected area before change are made.

(iv) Production of excessive frictional heat with conveyors, brakes, and bearings should be avoided by good installation and proper maintenance.

The production of frictional sparks, especially by metal-to-rock contact as with face-cutting, drilling, continuous mining, and longwall powerloading leader, that the incidence of ignition in thin seams resulting from picks striking the floor is much greater than from picks striking the roof due to the cutting zone at the floor being most difficult to ventilate, and that the risk of ignition is least with L-shaped outboard arm ranging drum shearer loaders and maximum with conveyor-mounted trepanners [15].

The risk of frictional ignitions by ripper-type continuous miners can be prevented or minimized by modified bit design and using carefully-engineered back sprays on the ripper drum with adequate spray pressure at the nozzle applied directly at the coalcutting bits.

Owing to the dominant role played by friction as an ignition source, one may divide the mine into abnormally high-suspect zones or areas of frictional ignitions, in which an intensive programme to eliminate frictional ignitions may be undertaken.

(v) Use of materials constructed of light alloys should be prohibited to eliminate incendive sparking.

(vi) Spontaneous heating of coal should be controlled by proper planning of mine development as well as coal extraction, good ventilation system, and inspection. Mine fires should be detected in their early stages.

Figure 2.7 Methods of charging a shothole with water stemming.

(vii) Blasting with explosives should be restricted to a minimum. Immediately before firing a shot or a group of multiple shots and after blasting is completed, examination for methane should be made. Mechanical winning and heading machines and blasting devices such as Hydrox, Cardox, and Airbreaker shells may be used as alternatives to ordinary explosives for breaking down coal. Pulsed-infusion shotfiring in combination with deep-hole water infusion may also be employed for getting coal without machine-cutting. Methods of charging a shothole with water stemming are shown in Fig. 2.7.

(viii) Welding, flame-cutting, grinding, and soldering operations must be done only after taking extreme precautions. Examinations for methane should be made immediately before and periodically, during such work.

(ix) All mobile diesel-powered equipment must be inspected and maintained in the approved condition in accordance with the instructions furnished by the manufacturer. Particular attention should be paid to the flame arresters

2.1.6.3.1 *Methane drainage techniques*

Methane drainage can be accomplished by a variety of underground and surface techniques for control of methane during mining or in advance of mining. For the success of any drainage technique selected, the drainage system must be properly designed and effectively controlled.

On the mine surface, the drained methane gas is usually conducted away to be burnt in a boiler house for power generation.

2.1.6.3.1.1 *Underground techniques*

The major underground techniques for methane drainage use cross-measure boreholes, superjacent headings, gob gas cavities, and horizontal boreholes.

Cross-measure boreholes

This is by far the most common method of methane control on European longwall faces. Cross-measures boreholes are drilled from the gate roads of advancing longwall faces into the adjacent strata above and/or below the seam being worked over the waste area and close to the working face (Fig. 2.8), usually using special underground rotary drills (Fig. 2.9). Methane is drained by applying suction pressure to the hole, and the methane–air mixture is drawn into a pipeline which may extend to the surface. The diameter of the boreholes varies between 65 and 120 mm. The spacing of the boreholes is important as too wide a spacing will allow a significant proportion of gas to escape into the mine workings, while too close a spacing will involve unnecessary and expensive drilling. The usual distance between the holes is 10 to 30 m when the zones of influence of the individual holes just meet. The optimum spacing is determined by a certain amount of trial-and-error drilling. With a properly designed borehole pattern 50 to 80 per cent of the total methane emission can be captured. The boreholes are drilled at angles varying from 30° to 60° to the bedding plane. The length of holes may vary between 25 and 40 m, exceptionally reaching 150 m. Holes drilled at right-angles to the line of the gate roads have been found from experience to be more successful than holes angled towards the face.

Gassy conditions
Very gassy conditions

Figure 2.8 Methane drainage through cross-measure boreholes.

The efficiency of the borehole-to-strata seal (standpipe seal) is of vital importance since it not only affects the purity of the gas but also the zone of influence of the boreholes and the effectiveness of the drainage system.

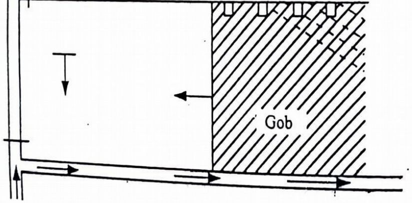


Figure 2.10 Methane drainage with double entry retreating longwalk

The drainage of gas is normally carried out through boreholes from the return gate or the tail gate but drainage from the main gate is also sometimes practised when gassy and very gassy seams occur in the floor of and close to the seam being worked to prevent methane from entering the air stream in the vicinity of electrical apparatus. Drainage of floor strata will permit extraction of a lower seam by retreating longwall. As a rule, boreholes at a distance of 10 to 20 m behind the face begin to yield gas, the yield reaching a maximum from a borehole at a distance of about 40 to 75 m from the face. The yield of gas from a borehole stops at a distance of 100 m behind the face.

however, methane entering the mine opening can be controlled (Fig. 2. 10).

Superjacent headings (Hirschbach method)

This method has been used in the highly gassy mines in the Saar Coalfield, Germany. It consists of driving a heading or roadway in the general strike direction in the strata 20 to 30 m above the coal seam and over the panel to be mined (Fig. 2.11). The roadway is sealed by stoppings to form a chamber and methane is drained from the induced fracture system due to extraction and drawn off through the seal using vacuum pressures of up to 3 m WG. When the nature of the strata allows, boreholes may be drilled in the floor.

In order to reduce costs, old roadways in previously worked-out seams can be used for this purpose.

With retreating longwall mining, drainage by use of superjacent headings is the best method especially in very gassy seams and seams liable to spontaneous heating. The method has a further advantage that there is clear separation between the drainage and production operations.

Horizontal boreholes in advance of working sections

Horizontal drainage holes drilled into a seam in advance of a panel or section (Fig. 2.12) can effectively lower methane emission levels at the working faces by machines to reduce gas emission into them.

Horizontal boreholes from shafts

The US Bureau of Mines successfully demonstrated that large quantities of methane can be drained through long horizontal boreholes drilled on a radial pattern into a virgin coal seam from the bottom of an air shaft (Fig. 2.13) long before the shaft is needed for ventilation. The advance drainage decreases the gas pressure within the seam over a large area around the shaft and reduces the methane load in the ventilating air. The successful application of this technique of de-gassification, however, depends on the *in situ* permeability of the coal seam and the hole direction relative to the cleat. This technique has been successfully applied in some mines in the USA.

Water infusion

Water infusion of a coal seam having a good fracture permeability and a favourable cleat orientation can reduce methane emission at the working face by 80 to 90 per cent by blocking the migration of methane towards the face [34].

Goaf/gob drainage

In very gassy mines, large quantities of methane may flow into the gob from the fractured strata above the coal seam. The large volume of the accumulated gas

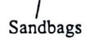


Figure 2.14 Methane drainage through gas cavities with caving of goaf.

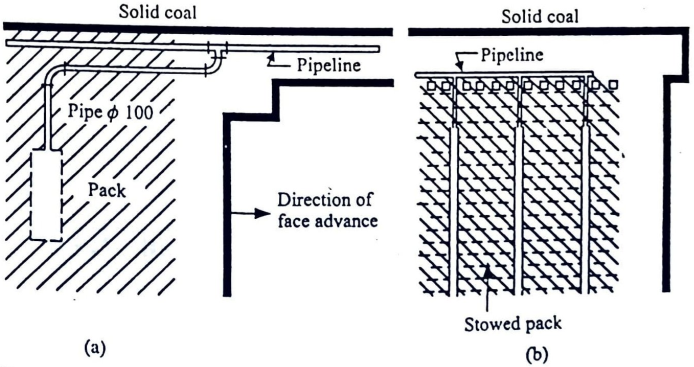


Figure 2.15 Methane drainage through pack-cavities.

control and borehole surveying as well as

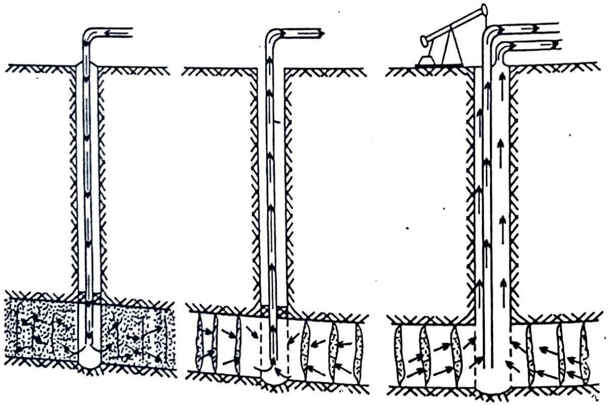


Figure 2.16 Methane drainage through surface vertical boreholes into virgin coal seam.

Figure 2.18 Methane drainage through surface vertical boreholes into goaf area.

Vertical boreholes into gob areas (Fig. 2.18)
Gob drainage with vertical boreholes has met with varying degrees of success in the USA and the erstswhile USSR coal mines. The holes are drilled ahead of mining to intersect the seam or terminate in the immediate roof above the seam. When the mining face passes the hole, methane is released into the gob from the fractured strata and flows to the hole to be drawn to the surface. The bottom 30 to 80 m of the cased hole is slotted or perforated to allow the hole to stay open when the roof caves. Experience has shown that large quantities of methane can be removed this way and that methane emission underground is reduced by more than 50 per cent. The method can be economical in shallow mines.

2.1.6.3.2 *Control of gas outbursts*

In seams liable to gas outbursts, special precautions should be observed in their working . Pre-draining the methane at a controlled rate by advance boreholes, drilling large-diameter de-gassing and de-stressing boreholes in the seam, high-pressure infusion of coal with water or $CaCl_2$ solution (30 per cent), extraction

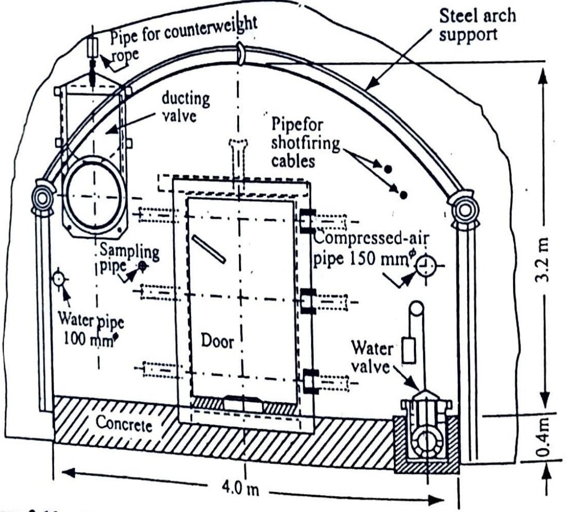

Pipe for counterweight rope
Steel arch support
ducting valve
Pipefor shotfiring cables
Compressed-air pipe 150 mm⌀
Sampling pipe
Water pipe 100 mm⌀
Door
Water valve
Concrete
3.2 m
0.4m
4.0 m

Figure 2.19 Blasting dam/stopping.

2.2.1 Development of coal-dust explosions

The exact ignition mechanisms of flammable dust–air mixtures are not clear. The view commonly held at present is that surface oxidation of coal-dust particles, partial de-volatilization of solid coal particles, ignition of the volaltiles–air mixture in the space between particles, and combustion of solid particles are conjointly responsible for the development of a coal-dust explosion. Experience has shown that a coal-dust explosion often develops in stages. The first stage of ignition is often a puff which is a sudden combustion of a part or the whole of the airborne dust marked by a very high temperature but without any dynamic effects. If the whole of the dust is consumed, the combustion of the dust–air mixture ends with the puff. On the other hand, if a part of the dust is consumed, the puff may eventually lead to an intense main explosion. During the transition from the puff to the explosion, there takes place an increase in pressure in the burning layers due to the transfer of the heat of combustion to the surrounding unburnt layers, mainly by radiation. When the pressure exceeds a certain value, a full-fledged explosion is developed in which the combustion takes place at such a rapid rate (velocity of propagation) that the whole of the heat liberated is transferred without loss to the surrounding unburnt layers. A pressure wave is developed which passes through the unburnt layers ahead of the explosion flame. Once the pressure wave is formed, the propagation of the explosion does not depend any more upon the

the particle size and the chemical composition of the dust, nature and intensity of the igniting source, time of contact with the source, turbulence of the dust cloud, uniformity of dust dispersion, purity of the sample, and the composition of the atmosphere, specially the methane and oxygen contents. USBM researches on Pittsburgh and Pocohontas Seams coal dusts [36] showed that the lower limit is independent of the particle size for all sizes below some threshold or characteristic value above which the limit increases significantly with increase in particle size until a critical size is reached above which the dust cloud is non-explosive at any concentration or entirely non-ignitable at any reasonable temperature (Fig. 2.20). Bituminous coals have a lower limit of flammability which is variously reported to vary widely between 30 and 300 g/Nm³ [37].

The upper limit of flammability or upper explosive limit is indefinite and has not been determined because of experimental difficulties of maintaining dust clouds of high concentrations. The upper limit is of little practical utility as it is unlikely ever to be reached under mining conditions.

As with methane–air mixtures, explosions of limit dust–air mixtures are weak. Curves obtained by plotting explosion pressures and rates of pressure rise against concentration show that the explosion pressure and rates of pressure rise are at a minimum at the lower limit of concentration and then gradually increase until

2.2.3 Ignition of coal dust–air mixtures

Ignition of dust–air mixtures is not so rapidly brought about as with gas–air mixtures. In general, a rather larger source of heat is required. The ignition temperature of a coal-dust cloud depends on the rank of the coal, nature and intensity of the ignition source, particle size, moisture and ash contents, and oxygen concentration. It is usually determined in the laboratory as the lowest furnace temperature at which a given size of dust ignites when dispersed into the furnace of a standard test apparatus (1.2–*l* Hartmann furnace in the U.S.A) as a uniform cloud. The temperature so measured is also referred to as the minium thermal auto-ignition temperature (AIT) as the explosion of a dust cloud is caused by its intrinsic reactivity at an elevated temperature. The ignition temperature decreases as the rank of the coal decreases from the anthracites to sub-bituminous coals. It varies between 900° and 600°C for relatively low-ash coals. Moisture in dust particles and in the surrounding atmosphere raises the ignition temperature of the dust because of the heat absorbed for heating and vaporization of the moisture. With decreasing oxygen concentration, the ignition temperature increases. Figure 2.21 shows the minimum auto-ignition temperature as a function of particle size for Pittsburg and Pocohontas coal dusts obtained using the 1.2–*l* Hartmann furnance [36].

2.2.4 Factors affecting development of coal-dust explosions

The flammability or explosibility of coal dust–air mixtures has been the subject of many investigations carried out in laboratories, surface experimental galleries, and experimental coal mines in many coal mining countries of the world. The work in experimental coal mines centred around the origin, propagation, and control of coal-dust explosions.

The following physical and chemical factors exercise an important influence on the flammability of coal dust–air mixtures and propagation and control of coal-dust explosions in mines.

Particle size

Thermal ignitability studies on coals of different volatile contents have shown that coal-dust particles up to 750 to 1000 μm [36] take part in explosion, depending on the rank of the coal, nature and intensity of the ignition source, and oxygen concentration and that fine particles control the ignition, violence, and speed of flame propagation. Dangerous are the particles which lie between

coal, and other parameters. The dust cloud must be dense immediately surrounding the ignition source so that one cannot see through it. As the dust concentration in mine air currents is not sufficient to propagate an explosion, the explosion must stir up the deposited dust along its path to create a cloud for its propagation.

Figure 2.22 shows a typical thermal ignitability curve for a coal dust showing the relation between the ignition temperature and the coal-dust concentration. It represents the boundary between ignitable and non-ignitable mass concentrations at a given thermal ignition temperature. It will be seen that the minimum value of the ignition temperature is approached asymptotically at the higher dust concentrations.

Volatile matter content

The flammability of a coal dust depends greatly on its combustible volatile matter; it increases with increasing volatile content. The volatile matter is usually calculated on a dry, ash-free basis using the formula.

$$\%\mathrm{VM} = \frac{\%\ \mathrm{VM\ (from\ analysis)} \times 100}{100 - \%\ \mathrm{ash} - \%\ \mathrm{moisture}}$$

Figure 2.22 A typical thermal ignitability curve for a bituminous coal dust in air.

As the right-hand side of the above formula may be written as

$$\frac{\%\,\mathrm{VM}}{\%\,\mathrm{VM} + \%\,\mathrm{FC}},$$

the percentage of volatile matter on dry, ash-free basis is sometimes called the volatile-combustible ratio or simply the volatile ratio of coal.

Several investigations in laboratories, surface experimental galleries, and experimental coal mines on the flammability of coal dusts had shown that coal dusts with VM $<$ 10 per cent are non-flammable, those with VM $=$ 10 to 14 per cent are less flammable, whilst those with VM $>$ 14 per cent are highly flammable. The increase in flammability, however, is different for different coals. For low-volatile coals, the flammability measured by the amount of incombustible required to prevent explosion flame propagation, increases almost linearly with increasing volatile ratio. But above about 25 per cent, the volatile ratio in the range of medium- and high-volatile bituminous coals, the flammability rises only slightly with increasing volatile ratio. From statistics of coal-dust explosions in the coal mines of Germany and the UK, it was noticed that explosions did not take place with coal dust containing less than 20 per cent volatile matter (daf).

Percentage of ash
An increase in ash content or presence of inert foreign material reduces the ignitability or flammability of coal dust because of heat absorption. Stone-dusting in coal mines is a practical application of the use of inert dust to prevent an explosion from taking place.

Percentage of moisture
Moisture in dust particles raises the ignition temperature of the dust. It exerts a cooling effect because heat is absorbed during its heating and vaporization, thereby reducing the energy available for ignition of the dust cloud. It also tends to wet and agglomerate the fine particles of dust, reducing their dispersability. Coal dust loses its dispersability with a moisture content of 25 to 30 per cent. The effect of moisture on flammability is unimportant below about 10 per cent.

Oxygen concentration
Variations in oxygen concentration affect the ease of ignition of dust clouds and the explosion pressures. With decreasing oxygen concentration, the ignition energy required increases, ignition temperature increases, and maximum explosion pressure decreases.

damaged electric cable.

Percentage of firedamp in mine air

The presence of firedamp in air in percentages less than its own lower limit reduces the lower explosive limit of coal dust by replacing the coal on a weight thermal basis. One per cent CH_4 (by volume) in $1m^3$ of air is equivalent to about 12 g of coal dust. The flammability of coal dust increases almost directly in proportion to the percentage of firedamp (Fig. 2.24) [39].

Turbulence

As the combustion of dust takes place at the surface of the dust particles, the rate of reaction, therefore, depends on how intimately the dust and oxygen are mixed. Turbulent mixing of dust and air will result in more violent explosions than those caused by ignition in relatively quiescent mixtures.

Surrounding conditions

The size, shape, constrictions, obstructions, branching, length, and nature and condition of the surfaces of mine workings exert an important influence on the development of coal-dust explosions as they increase or decrease the progress of a flame by holding or releasing the pressure.

$P_{initial}$

where P_{max} is the maximum explosion pressure (absolute) during an experiment, ΔP (ignitor) is the pressure rise due to the ignitor ignition source by itself and $P_{initial}$ is the pressure prior to ignition (~1 atm absolute).

Rate of propagation of explosion flame or flame velocity

Velocities of propagation of explosion flame up to 2000 m/s or more (usually 200 to 300 m/s) are encountered in mines. In a roadway of constant cross-section which is more or less uniformly dusty, the velocity of flame propagation and explosion pressure increase with the distance from the seat of the explosion. The velocity is greatly affected by sudden variations in cross-section of the mine workings through which the explosion passes, the amount of dispersed stone dust, and the pressure release into adjacent areas.

The average burning velocity of a coal dust–air mixture is given by the ratio

$$\frac{\text{Average flame speed}}{\text{Expansion ratio}} = \frac{\text{Max.travel distance}}{\text{Time to peak pressure}} \times \frac{1}{\text{Expansion ratio}}$$

The expansion ratio is the density of the unburnt mixture divided by the

$$\text{Ignition severity} = \frac{(\text{Max. Explo.Press} \times \text{Max. Rate of Press. Rise})\ \text{Sample dust}}{(\text{Max. Explo. Press} \times \text{Max. Rate of Press. Rise})\ \text{Standard coal dust}}$$

$$\text{Explosibility index} = \text{Ignition Sensitivity} \times \text{Explosion Severity}$$

Depending on the explosibility index, the coal-dust explosions may be classified as follows [41]:

Type of explosion	Explosibility index
Weak	< 0.1
Moderate	0.1–1.0
Strong	1.0–10
Severe	> 10

Index ratings based on the above have, however, fallen out of favour.

In Germany, a coefficient of explosibility is calculated from data obtained on coal-dust explosion in large closed test vessels as

$$k_{ex} = \sqrt{(\text{Max. rate of pressure rise} \times \text{avg. rate of press.rise})}\ \text{bar/s}$$

The explosibility factor or simply the explosibility of a coal-dust deposit is given by

Formation of coke crust and soot deposits

In a coal-dust explosion, its pressure wave and flame radiate intense heat which can melt metals and convert organic matter into coke and ash. Owing to insufficient oxygen, only a part of the coal dust is completely burnt into ash whilst the remaining dust gets coked and deposited as characteristic crust or globules on roadway supports and on the sides and roof of the roadways through which the explosion passes. The presence of visible traces of coke crust after an explosion depends on the caking properties of the coal. Bituminous coals yield larger coke crusts than non-coking coals which yield granular crusts. Experience with large explosions in mines has often shown that the coke crust is invariably formed on the reverse or sheltered side of the roadway supports or projections (Fig. 2.25) and that it is found on the 'windward' side of the supports or projections near the seat of the explosion. In small explosions with lower velocities, on the other hand, the coke crust is formed on all sides of the supports. Experiments in the USBureau of Mines Experimental Coal Mine [42] showed that an explosion flame can traverse an area without producing coke and that as the explosion flame velocity increases, the amount of coke produced decreases.

Soot is deposited in and beyond the area traversed by the explosion flame. Soot deposits are generally thicker in the explosion area. Soot may be deposited as a continuous layer on horizontal surfaces or form a lacy network or stringers at the roof and on vertical surfaces.

(9) The ignition of a coal-dust explosion even with the strongest igniting source requires at least 100 ms, while with a firedamp explosion only 1 to 2 ms is required [43].

(10) The velocity of propagation of coal-dust explosions is generally higher than that of firedamp explosions.

2.2.7 Dust explosion test apparatus

Various types of apparatus for studying the flammability behaviour of coal dust–air mixtures have been described in the literature on coal-dust explosions. The different types of test chambers used and varying procedures adopted make it difficult to compare results obtained by them. Over the years, larger and more spherical test chambers have been developed with uniform dust dispersion and adequate strength of the ignition source for hard-to-ignite dusts and dust-inhibitor mixtures which would give dust explosion data that correlate well with large-scale tests and experimental mine results. In these test chambers, it is possible to measure the lower and upper limits of flammability, explosion pressures and rates of pressure rise, explosion temperatures, minimum ignition energies, minimum oxygen concentrations for flammability, and amounts of inhibitors necessary to prevent explosions.

consists essentially of an electrically-heated vertical cylindrical tube (Alundum 3.9 cm × 23 cm) furnace through which a weighed amount of coal dust is projected downwards as a uniform cloud by a controlled blast of air directed at the dust sample placed in a small chamber at the top of the tube. The lowest temperature of the furnace at which flame issues out of the open bottom end of the tube gives the thermal ignition point.

The Godbert–Greenwald furnace suffers from the drawbacks that the dust cloud is not completely confined within the furnace and its concentration is not accurately known, short exposure time of the dust cloud in the high-temperature region, and uncertainty as to the vertical temperature uniformity.

An improved version of the above furnace is the 1.2–*l* cylindrical Hartmann ceramic furnace or the bomb developed in the 1930s which became a standard dust explosibility apparatus in the USA and which generated the largest amount of dust explosion data. This apparatus, however, has serious deficiencies such as non-uniform dust dispersion, inadequacy of the ignition source for many dusts, and lack of agreement with large-scale tests. A 8–*l* flammability chamber was developed by the US Bureau of Mines for coal-dust and gas explosion studies. Figure 2.27 gives a schematic diagram of the chamber and the air-dispersion tank. The instrumentation includes a pressure transducer, an oxygen sensor, and an optical dust probe. The pressure transducer measures partial

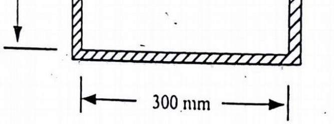


Figure 2.27 USBM 8–*l* Flammability chamber.

pressures when mixing gases and also explosion pressures after ignition. The oxygen analyzer uses a polarographic method to measure the partial pressure of oxygen. The optical dust probe is used to study the dust dispersion by measuring the attenuation of near-infrared light at a wavelength of 0.95 μm. The standard ignitor is an array of four electrically. activated pyrotechnic matches which deposit their energy (140 J) in about 0.05 s. The conical dispersion cup significantly improves the effectiveness and reproducibility of the dust dispersion.

The normal procedure is to spread a measured mass of dust uniformly around the disperser cone, partially evacuate the chamber to about 0.2 to 0.3 atm, disperse the dust by an air impulse injected by the air-dispersion tank (pressurized to 5 atm), raising the chamber pressure to 1.0 atm, and energise the ignition source, allowing 0.1 s after the duration of the air impulse (0.2 s).

The 8–*l* chamber produced much valuable data on the flammability of coals and other dusts. Owing to the limitation of the maximum ignition energy that could be used and use of test chambers up to 1m^3 in European countries, the Bureau of Mines developed a 20–*l* explosibility test chamber (Fig. 2.28) which can be used for explosibility testing of dusts, gases, and their mixtures using stronger ignition sources at lower and higher initial pressures. The data obtained have shown good agreement with data obtained from full-scale tests in an experimental mine [28].

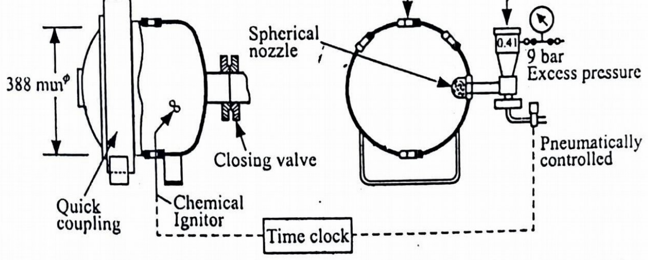


Figure 2.29 German 40–*l* explosibility test chamber.

At the Bergbau-Versuchssrecke (Mining Experimental Gallery), Dortmund Derne, Germany, test chambers of 1.2–*l* to 30 m^3 capacity are available for coal-dust explosibility studies. For routine testing, 1m^3 and 40–*l* capacity test chambers are used. Figure 2.29 shows the 40–*l* test chamber [44]. The dust to be tested is kept at a pressure of 9 bar(g) in a dust chamber outside the test vessel. It is dispersed after the opening of the solenoid valve and, after a delay of 170 ms, a chemical

In practice, the causes that bring about ignition of flammable dust–air mixtures are very similar to those operating in the ignition of flammable methane–air mixtures, but the explosion hazard, in general, is rather greater as coal dust is a normal accompaniment of the coal winning process and can be easily raised into the mine air as a dust cloud. For a coal-dust explosion to take place in mines, two conditions must be fulfilled. The dust must be present as a dense cloud and a source of ignition in the form of a flame must be present.

The various causes of direct ignition of a dust cloud can be classified as

(a) naked flames
(b) friction
(c) electric sparks
(d) firedamp explosions

Naked flames

A naked flame is the easiest means of igniting a dust cloud as the source of heat is of considerable size and a larger part of the dust cloud can be heated.

significant that most firedamp explosions do not develop into coal-dust explosions due to their failure to raise a sufficiently dense dust cloud.

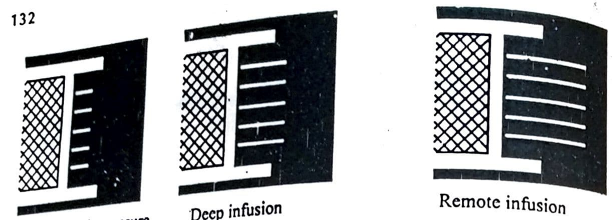
2.2.9 Prevention and suppression of coal-dust explosions

It is well-known that it is much easier to prevent a coal-dust explosion being initiated than to suppress one.

The measures for prevention and suppression of coal-dust explosions can be divided into the following three groups:

2.2.9.1 Measures which prevent or reduce formation and dissemination of coal dust underground

The elimination of coal dust in mines would be the logical means of eliminating coal-dust explosion hazard, but unfortunately, this is virtually, impossible. Much can, however, be done to reduce the formation, distribution and accumulation of coal dust in mine workings.



Short-hole high-pressure infusion

Deep infusion

Remote infusion

Long-hole high-pressure infusion

Roadway
Borehole

consumption of considerable amounts of water and energy and the possibility of wetting a whole section of the seam.

(ii) Wet winning of coal using wet pneumatic picks where used.

(iii) With machine-cutting by means of coal cutters, using sharp picks of a suitable type, selecting optimal cutting and travelling speeds of the machine, using gummer and wet-cutting; with continuous miners by using scrubbers on them.

(iv) When blasting in coal, use of stemming cartridges or ampoules of calcium chloride powder containing 82 to 85 per cent $CaCl_2$ and 15 to 18 per cent water of crystallization reduces considerably the dust produced by shotfiring due to the formation of a large number of droplets as a result of distribution of hygroscopic salt particles in the fume cloud containing a considerable amount of water vapour, which bind the dust.

(v) With power loading using the conventional shearer loaders, by not using too many cutting bits which should be sharp and of a suitable type and bit lacing, selecting a suitable drum design with vane angle greater than 12°, operating at optimal rotational speed of the drum by using a two-speed

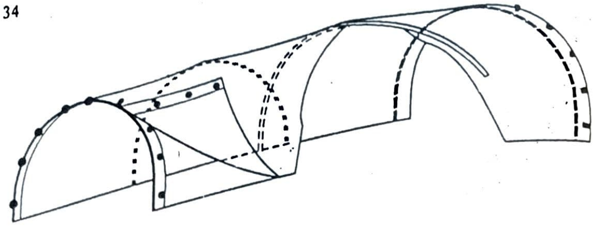


Figure 2.33 Covering hood for belt conveyors.

(vii) With belt conveyors;
  (a) covering them along their length with a hood (Fig. 2.33);
  (b) complete shrouding or boxing of the conveyor for a certain distance, before the entry and after the exit when passing through air-locks with some form of seal (suspended conveyor belting) to allow passage of materials;
  (c) having properly designed and installed chutes at transfer and loading points. It is most advantageous to hood or enclose the transfer points; and

  b) incorporating a dust collector in the auxiliary ventilation system in development drivages and headings;
  c) cleaning systematically and regularly main haulage roads and main return airways of deposited dust (three to four times a year) using transportable roadway suction apparatus;
  (d) cleaning regularly and systematically at and near transfer and loading points;
  (e) installing skiphoisting in upcast shafts; and
  (f) locating dry coal preparation plants far away from downcast shafts (not less than 80 m).

(xv) Selecting a method of coal winning with which dust production is the least. Winning by coal ploughs produces much less dust than by shearer loaders.

(xvi) Controlled caving of roof coal in very thick seams mined by the sub-level caving method using close-fitting shields.

(xvii) Consolidating the floor dust to prevent it from being raised.

2.2.9.2 Measures against ignition of dust accumulations

coal dust was recognized. For water to be effective, it must be present in sufficient quantity and be mixed intimately with coal dust. The percentage of admixed water (by weight) required to prevent propagation of a coal-dust explosion depends on the fineness of dust, strength of the ignition source, volatile content of coal, and the amount of stone dust present. It should not be less than 50 per cent. In practice, coal dust is wetted over a length of roadway at least 8 m near a working face by washing down excessive deposits of it on the side and roof surfaces to the floor by means of hoses thereby converting the dust into mud. The frequency of water washing depends upon the dryness of dust and frequency of blasting. Water spraying using water-spray nozzles installed not more than 10 to 20 m from working faces have been regarded as effective means of combating initiation of coal dust and firedamp explosions by blasting. To increase the wettability of coal dust which neither adsorbs nor absorbs water readily, a wetting agent may be added to the water.

The value of stone dust application is due to the following facts:

(a) When coal dust is mixed with enough inert or incombustible dust, it will not explode or assist in propagation of an explosion. An addition of about 40 per cent stone dust causes a 50 to 90 per cent reduction in flame velocity.

(b) When incombustible dust is raised into an explosion flame in a sufficiently dense cloud, it gets heated up taking away heat from the flame. Greater the amount of stone dust in the flame, greater is the cooling effect. In the case of coal-dust explosions, the incombustible dust also serves to shield or blanket the coal-dust particles from the heat of radiation so that they do not take part in combustion any more and also hinder diffusion of oxygen and gases into and from the burning coal particles.

As coal dust is constantly produced as a result of mining operations and gets deposited by ventilating currents in mine workings, one application of stone dust will not suffice to keep the mine workings immune from explosions. Stonedusting

2.2.9.3.2.1 *Characteristics of suitable stone dust*

All stone dusts used for stonedusting and other purposes in mines must fulfil certain requirements in respect of fineness, dispersability, combustible matter, and effect on health. Different countries have prescribed different specifications for a suitable stone dust (Table 2.5).

A suitable stone dust should not be hygroscopic so that it does not cake when not directly wetted by water from the strata. It should be readily dispersable into the air when blown by the mouth or by a suitable appliance, and be soluble in the fluids of the lungs. It should also be as light in colour as possible, preferably white, so that it makes it easier to detect deposition of a thin surface layer of coal dust where coal dust is building up.

Table 2.5 Specifications of stone dust in some countries

	Germany	USA	UK	India
Fineness	100% through sieve of 144 mesh/cm^2 and	100% through 20-mesh per linear inch	100% through BS 60-mesh sieve and at	100% through 60-mesh and at least

Table 2.6 Relative efficacy of inhibitors in inerting coal dust accumulations

	% Inert content required in	
Inhibitor dust	Experimental mine	8-*l* Chamber
$NH_4H_2PO_4$, ABC	25–30	28–12
NaCl, BCD	40–45	50–55
KCl, Super K	40–45	55–60
$CaCO_3$, stone dust	70–72	65–70
$KHCO_3$, Purple K	70–75	77–82

2.2.9.3.2.2 *Stone dust application*

As deposition of coal dust takes place continuously, stonedusting should be done in the right place in such a manner and at such intervals of time as will ensure that the mine dust on the floor, roof, and sides throughout will always consist of a mixture containing not less than the prescribed minimum percentage of incombustible matter. Excessive or blanket stonedusting of the floor of mine workings can never compensate for deficiency of stone dust on the sides and roof.

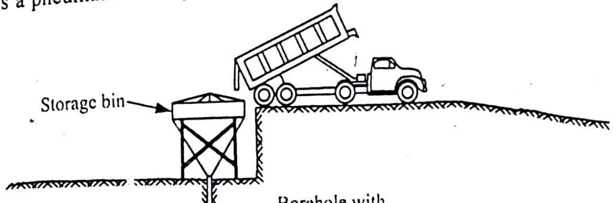
Before applying stone dust for the first time, all accumulations of coal dust, Mine Door Co., Ohio, USA. They are equipped with 5 to 20 hp electric motors and are available as skid-, track- or rubber tyre-mounted units (Fig. 2.34). Their minimum heights are 400 mm on skids, 558 mm on rubber tyres, and 635 mm above rail on wheels. The skid-mounted machines may be mounted on shuttle cars, belts, or mine cars. The tyre- and track-mounted units may be of portable, mobile, or towed type. Experience has shown that the tyre-mounted unit towed by a battery-operated tractor provides the maximum flexibility, manoeuvrability, and range. In mine workings containing compressed-air lines, compressed-air operated tank-type dusting machines may also be employed.

Mechanical dusting should be carried out only when the workplaces to which the dust is carried by ventilating currents are free of workmen. It must be carried out between shifts, on idle shifts or weekends, by gangs specially trained for the purpose.

Auxiliary exhaust fans when used to improve ventilation can also be utilized for stonedusting by introducing dust at the fan discharge.

In mines with extensive mine workings where the handling and distribution of stone dust would consume considerable number of manshifts, the so-called 'bulk stonedusting system' may be adopted with which the total capital invested can be realized within a short time. In the bulk stonedusting system as practised in the US mining industry, the stone dust is transferred from a surface storage bin into large-capacity bulk supply cars underground through a lined borehole (Fig. 2.35) by means of an aerator feeder. The bulk cars are transported to different sections of the mine where the stone dust is transferred to stonedusting machines by screw-type conveyors. In a shift, two stonedusting men can handle about 20t bulk stone dust including the filling of the bulk car at the borehole, transporting it to the section, filling the hopper of the dusting machine and stonedusting.

MSA developed a pressure tank automated stonedusting system (Fig. 2.36). It is a pneumatic conveying system composed of modular units which can be



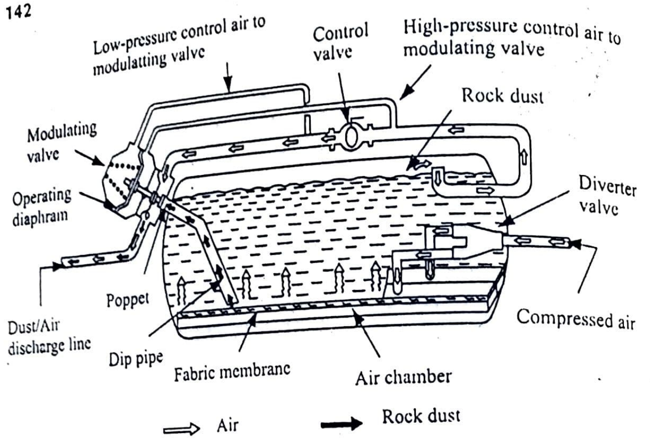

Low-pressure control air to modulatting valve
Control valve
High-pressure control air to modulating valve
Rock dust
Modulating valve
Operating diaphram
Diverter valve
Poppet
Compressed air
Dust/Air discharge line
Dip pipe
Fabric membrane
Air chamber
Air
Rock dust

Figure 2.37 Construction of a pressure tank fitted with a modulating value.

grouped to suit mine layout. A storage bin located on the surface supplies stone

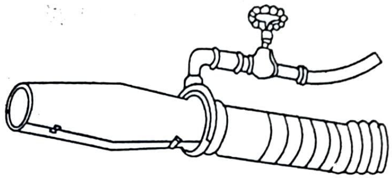


Figure 2.38 Wet nozzle.

(c) the rate of drying of the stone dust varies with the air velocity and relative humidity; with normal air currents and relative humidities below 80 per cent, the stone dust dries completely in one to three days and at humidities of 80 to 90 per cent, it dries in about a week; and

(d) wet stonedusting is less effective than dry stonedusting in arresting the propagation of an explosion.

2.2.9.3.2.3 *Testing of stone dust before use in mines*

Stone dust must be tested at the mine laboratory for fineness, purity, and dispersability.

Testing for fineness

From the consignment arriving at the mine, a gross sample of approximately 2

2.2.9.3.2.4 *Mine dust sampling*

The only certain means of controlling the adequacy of stonedusting is by taking samples of the mine dust from appropriate locations at regular intervals depending on the intensity of coal-dust production in the mine and testing them for their incombustible contents. Sampling procedures are laid down in mining regulations. A mine is divided into zones or sampling lengths which should be distinctly demarcated, and a representative sample is taken from each zone at such intervals of time as may be prescribed. The length of a zone should not be less than 50 m. The length, number and location of such zones should be clearly shown on a plan called the sampling plan. The mine workings and places in them from which it is not necessary to collect samples are the main shafts, production faces, development headings for opening up of longwall faces, loading and transfer points, and workings and places which are naturally wet or treated with dust-binding salts. The results of the analysis or tests should be recorded in a special book (dust register) kept at the mine for the purpose, which shall also show the place (location in the zones), the date of last stonedusting at the sampling place, and time of sampling. In general, there should always be liaison between the stonedusting and sampling operations to avoid the possibility of sampling

(7) A carrying canvas sack or sectionalized carrier box to carry 30 to 36 samples.

The collection of a representative sample from a zone is usually by the method of strip sampling by which the mine dust is collected from a succession of transverse strips as nearly as may be of equal width (100 or 150 mm) taken at such number of places or such distance from one another over such length of zone as may be prescribed by regulations. When collecting a sample, the sampler should walk against the ventilating current.

At any sampling section, two kinds of samples are collected: the floor sample and the side-roof sample. The floor sample comprises dust collected to a maximum depth of 25 mm on the floor from side to side provided that loose material is found to that depth, while the side-roof sample comprises dust collected to a depth of 5 or 6 mm as near as may be on the roof and sides. The surfaces of fixed structures are considered to be part of the roof and sides.

The floor and side-roof samples may be combined into a single sample and a single representative sample (250 to 500 g) obtained after mixing, quartering, drying in the air if necessary, and passing a portion through the prescribed sieve sampling) or separate representative samples obtained from reduced. Portable coal dust/rock dust analysers for on-the-spot evaluation of the adequacy of stonedusting have been developed in the USA which will indicate the incombustible content of mine dusts over a range of 50 to 100 per cent incombustible within ± 2 or 3 per cent of the chemically determined incombustible content for 80 per cent of the standard samples with prescribed specifications.

Chemical analysis

For routine dust analysis, chemical analysis consists of determination of the moisture and ash contents.

Moisture content

A weighed quantity (1 g) of the representative sample is dried at a temperature between 105° and 110°C. The loss of mass represents the moisture.

$$\%\text{Moisture in sample} = \frac{\text{Mass loss of moisture}}{\text{Mass of dust sample}} \times 100$$

Ash (incinerated residue) content

In the high tempremature incineration method, the residue remaining after moisture determination is heated to a temperature of 925° ± 25°C in a muffle furnace for one or two hours until it no longer loses mass and the mass of the incinerated residue determined.

Colorimetric methods

There are two practical rapid methods:

(1) Usual colour comparison method; and
(2) Colour measurement method using a reflectometer (instrument sorting).

In the visual colour comparison method, the colour of a representative sample is compared in the mine with that of standard dust mixtures prepared from actual mine dust samples and known to contain definite percentages of incombustible matter. To eliminate underground all samples that contain 80 per cent or more incombustible matter which would mean a considerable saving in effort, only a single standard dust mixture containing 80 per cent incombustible need be used.

In the colour measurement method by reflectometer, the light-reflective properties of the surface of mine dust samples are measured by means of photo-electric cells. The reflectometer must first be calibrated for different mines and seams by conducting tests on standard dust mixtures for each mine and seam and plotting the relation between the inert content of the dusts as obtained by chemical analysis and reflectometer readings. From the calibration curves, the percentage of inert content in an actual mine dust sample corresponding to the reflectometer reading obtained can be read off. The reflectometer can be used to sort dusts having an inert content between 50 and 80 per cent.

To use the apparatus, 25 ml of methylated spirit is first run into the flask and 25 g of the dust sample to be tested is then poured into it. After inserting the measuring stem into the vaselined neck of the flask, the flask is vigorously shaken by hand for two minutes to remove all air bubbles. The other 25 ml of the spirit is then poured through the open end of the measuring stem and the reading of the level of the liquid in the stem noted. By reference to the calibration graphs obtained for standard dust mixtures for different seams, the incombustible matter in a dust sample in any seam can be found out from the reading on the graduated stem.

To safeguard against any discrepancy when using the volumeter method, one in every 50 samples should be check-tested by chemical analysis. The accuracy that can be obtained with this method is ± 4 per cent [50].

Combination of the colorimeter and volumeter methods

When an increased throughput of samples is to be achieved, a combination of colorimetric sorting and volumeter testing may be used. The samples are first sorted by the colorimetric method and those that are darker in colour than the standard dust are analysed by the volumeter method.

2.2.9.3.2.5 *Efficacy of stonedusting*

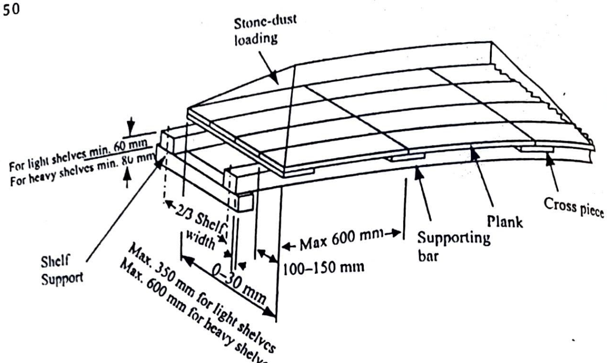
pressure of an explosion to release and disperse a mass of stone dust in the form of a thick cloud into the path of an oncoming explosion flame, thereby smothering the flame.

Barrier design

Experience has shown that of the various designs of barriers evolved and tested in Germany, Poland, and the USA, the shelf barrier, consisting of a number of dust-laden shelves independently supported transverse to and along the roadway in which it (the barrier) is installed, is the most practical design, being less difficult to install and maintain. It occupies a length of 25 to 40 m of the roadway.

Stone-dust barriers are installed in strategic areas of the mine such as roadways leading from shafts or their pitbottoms, in all level and inclined roadways including gate roads and development headings, and near roadway junctions.

Two designs of shelves are commonly used in mines, the German (Improved Dortmund-type) Shelf and the Polish Shelf. The German Shelf (Fig. 2.40) consists of several loose planks, maximum 60 cm long and 10 to 15 cm wide, which are abutted along their length with the ends supported by cross pieces which rest upon two long transverse bars with their ends resting loosely on rigid shelf supports. The Polish Shelf (Fig. 2.41) is composed of several short planks, 10 to 15 cm wide and 35 to 50 cm long, laid alongside one another in the direction of the roadway.


Stone-dust loading
For light shelves min. 60 mm
For heavy shelves min. 80 mm
2/3 Shelf width
0–30 mm
Max. 350 mm for light shelves
Max. 600 mm for heavy shelv
Shelf Support
Max 600 mm
100–150 mm
Supporting bar
Plank
Cross piece

The following rules govern the erection of stone-dust barriers of both types in German mines:

(1) A stone-dust barrier must carry not less than 400 kg stone dust per square metre of average cross-section of the roadway in which it is installed.

(2) At least a quarter of the total dust loading of any one barrier must be distributed on light shelves which must flank the heavy shelves of the barrier at either end. The heavy shelves must carry at least a quarter of the total dust loading.

(3) The distance between the adjacent shelves of a barrier must be as given under:

German shelves min. 1.50 m
max. 5.00 m

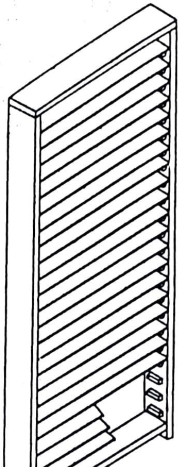
Polish shelves

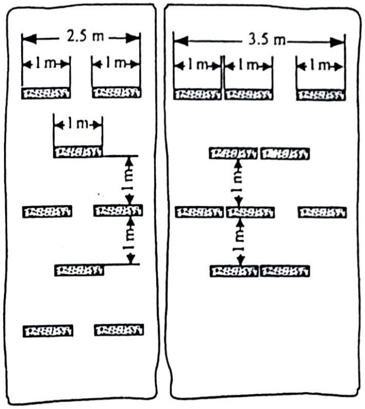
(14) The distance between barriers should not exceed 200 m in gate roadways of longwall faces and 400 m main roadways.

In the UK, the stone-dust barriers are of three types: the light, intermediate, and heavy barriers. Two types of Polish Shelves are used. The light shelf the width of which should not exceed 35 cm, has a dust loading not more than 29.8 kg/m shelf length, while the heavy shelf, the width of which should not be less than 35 cm and not more than 50 cm, is loaded with not more than 59.6 kg/m. The light barrier carries a total dust loading of at least 107.4 kg per square metre of average roadway cross-section and comprises all shelves of the light type. The intermediate barrier carries a dust loading of 195.3 kg per square metre roadway cross-section and comprises not more than one-third of the shelves of the heavy type. The heavy barrier carries a dust loading of at least 390.6 kg per square metre of average roadway cross-section comprising not less than one-third of the shelves of the light type.

The distance between the adjacent shelves are:

Light shelves	min. 0.91 in max. 2.10 in





have shown that a time interval of 0. 1 to 0.2 s [40] is sufficient enough to create a flame-quenching cloud.

In practice, stone-dust barriers may fail under any one of the following circumstances:

(1) When the barrier itself lies in a flammable firedamp–air mixture or firedamp occurs as a roof layer;
(2) When flame velocities are high (exceeding about 500 m/s), as when a dust explosion is initiated by a powerful firedamp explosion or is assisted by firedamp during its propagation;
(3) When the barrier is located less than 40 to 60 m or far from a face or other potential point of ignition, so that the dynamic pressure is less than the minimum required, which normally lies between 3 and 5 kN/m² ; and

(4) When an explosion is initiated by a weak ignition source and sufficient dynamic pressure is not built up.

The type, siting, design, installation, and maintenance of stone-dust barriers to be provided in longwall and bord and pillar workings in gassy seams of Indian coal mines in accordance with the sub-regulation (2) of Regulation 123C of the Coal Mines Regulations, 1957 (Appendix 1) are specified in the DGMS Circular No. 6 (legis) 75.

2.2.9.3.4 Explosion-proof stoppings

Unlike stone-dust barriers, explosion-proof stoppings localize firedamp or coal-dust explosions by isolating completely the area from the rest of the mine. An explosion-proof stopping must, therefore, withstand, without being damaged, the full explosion pressure and also bring about extinction of the explosion flame by starving it of oxygen. It is mainly used for localizing explosions in parts of a mine in which there is less likelihood of widespread propagation of an explosion such as development workings in coal and stone. In practice, explosion-proof stoppings are frequently erected in sealing off fire areas where there is danger great disadvantage of this process, however, is that it cannot be used where the relative humidity exceeds 80 per cent as no recrystallization takes place. In very dry conditions with relative humidity below 55 per cent, water evaporates so rapidly that frequent re-wetting becomes necessary, which makes the method uneconomical.

Calcium chloride/Magnesium chloride paste process

This process depends for its action not upon the formation of a crust by alternate processes of solution and crystallization but upon continued wetting of the coal dust by the hygroscopic action of calcium chloride or magnesium chloride paste which causes the dust particles to adhere or bind by extracting moisture from the atmosphere. The hygroscopicity of the calcium chloride or magnesium chloride alone will not suffice to wet the dust unless a wetting agent (1 to 1.5 per cent) is added to it. To improve adherence to rock surfaces, a gelatinous compound, magnesium hydroxide (2 to 3 per cent), is added to the calcium chloride or magnesium chloride solution which then assumes a pudding-like appearance. The percentage of calcium chloride or magnesium chloride depends on the prevailing climatic conditions. It must be such that the paste maintains its pudding-like state for as long as time a practicable. At higher temperatures and relative humidities, the required concentration of the solution will be less than at lower temperatures and relative humidities. Figure 2.44 shows the variation of CaCl concentration with the relative humidity at 30°C [17]

Like a stone-dust barrier, a calcium chloride- or magnesium chloride-paste zone will fail to suppress an explosion if it is filled with a flammable firedamp mixture or has a layer of firedamp at the roof.

On the European continent, greater reliance is placed on calcium chloride paste-zones than on generalized stonedusting or stone-dust barriers. Calcium chloride has a slight corrosive effect on rails, ropes, etc., which should be kept out of contact with the salt.

Calcium chloride powder process

The calcium chloride/magnesium chloride paste process suffers from a serious disadvantage that the paste, containing 20 to 30 per cent hygroscopic salt, must be transported in special containers to the place of use or pumped using high -pressure pumps where it is not possible to transport the containers. To overcome this difficulty, Chemische Fabrik Kalk GmbH, Germany, developed in 1965 the Montan Powder Process, in which a fine-grained high-percentage calcium chloride powder (80 to 85 per cent monohydrate $CaCl_2$, H_2O) containing a wetting agent (3 per cent) is used in place of the paste. Sodium chloride is added to serve as support material to promote adherence to clean surfaces and to extend the interval between applications. On account of its large surface area, the powder adheres to the roof and vertical surfaces. It the pithead, 1 t of powder (80 per cent $CaCl_2$) costs four to five times the cost of 1 t paste (20 to 30 per cent $CaCl_2$).

The advantages of the powder process over the paste process are that the storage and transportation of powder in plastic sacks are simpler; roadway roof, sides, and floor can be treated in one operation; it is less corrosive and can be used under all humid conditions generally prevalent in mines. A disadvantage of the powder is its high dispersability, which causes it to be carried over long distances. For this reason, the intake airways must be dusted when there are no men on the face. The addition of sodium chloride as support material makes it dispersible with difficulty.

2.2.9.3.6 Water barriers

Since the early 1960s, interest has revived in the use of passive water barriers, also called water-trough barriers, as an alternative to stone-dust barriers for suppression of coal-dust explosions in mines. Water has the following advantages over stone dust:

(a) its heat capacity is about five times that of dust;
(b) its efficacy is not affected by underground climatic conditions; and
(c) it is available in all mine roadways.

Contrary to expectations, troughs supported in the roadway cross-section with their edges resting on the transverse supporting frame have been found to shatter more easily than troughs supported on transverse bars. This is because the former are subjected to the full dynamic pressure and cannot be displaced as happens with the latter. Also, the use of loose lids to prevent evaporation of water does not in any way affect the distribution of water in the direction of the roof as the lid lifts itself off the trough.

Water barriers are at present widely used in German coal mines. Their design and installation procedures are laid down in the regulations. Depending on the method of support, two types of water barriers have been approved for use in mine roadways less than 4 m in height or less than the maximum width at the site of installation – Type I and Type 2. In Type 1 barriers, the troughs are supported with their total upper lip surface resting on transverse supporting frames while in Type 2 barriers, the troughs are placed over transverse bars. Figure 2.45 illustrates the prescribed method of installation of water troughs carried by a shelf in a roadway cross-section as well as the two types of approved trough construction.

The following rules govern the erection of water barriers in German mines:

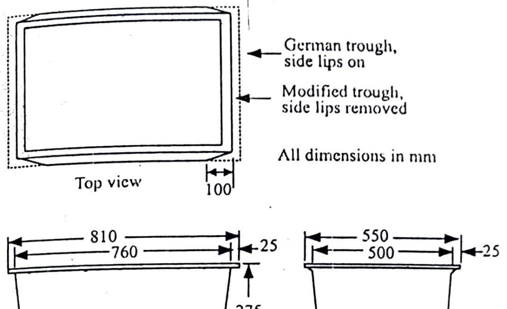
The water barriers must at all times contain a total quantity of at least 200

(ii) The distance between two shelves must be at least 1.2 m.

(iii) The space between lips of adjacent troughs carried by a shelf must not be greater than 1.20 m and the total space between troughs in a row of troughs must not exceed 1.50 m. A row of troughs is identified as a number of troughs supported across the roadways from side to side.

(iv) The distance between the outer troughs of a row and roadways side or rib must not exceed 1m. Where this distance exceeds 1 m, an additional trough containing at least 30 litres of water must be installed at the rib.

(v) The troughs must cover, at the site of installation, at least 50 per cent maximum roadway width for roadway cross-sections up to 14m², and 65 per cent maximum roadway width for roadway cross-sections exceeding up to14m² and 65 per cent maximum roadway width for railway cross-sections


German trough, side lips on
Modified trough, side lips removed
All dimensions in mm
Top view
100
810
760
25
550
500
25

probability that a few shelves will be at some optimum distance from the explosion origin to suppress the explosion flame. The distributed barriers were found to be effective in suppressing slow-moving to powerful explosions with static pressure of up to about 6 bar, dynamic pressures of up to 1.6 bar, and flame velocities between 30 and 500 m/s.

With distributed barriers, the quantity of extinguishing water is expressed in litre or kg per cubic metre of roadway volume. The quantity of water required, as found from trials in experimental mines, lies between 0.5 and 1 litre per m^3 and the number of troughs or containers required is no more than with the concentrated barriers.

The US Bureau of Mines developed three types of water barriers especially for use against slow-moving explosions. One barrier is a modification of the conventional German barrier which responds to dynamic pressures whereas the second and third types respond to rise in static pressures developed ahead of the explosion for their operation. All the three types were found to be effective against weak dust explosions.

2.2.9.3.7 Triggered barriers

Stone-dust and water barriers suffer from the disadvantages that they depend

polythene bag or polythene-covered container is dispersed by a detonating cord. In the second system, the extinguishing agent which may be water or powder, is contained in the disperser under a stored force (pressurized nitrogen gas or Halon 1301) and is dispersed when a detonator bursts the sealing disc or diaphragm of a quick-release valve of an inverted high-pressure steel cylinder containing the suppressant water or powder and nitrogen at a pressure of 30, 60, or 120 bar(g).

It is essential that the extinguishant is discharged at a high rate and that the time interval between the onset of discharge and flame arrival does not exceed the time necessary for full discharge.

The relative effectiveness of the extinguishing agents tried has been found to be as follows [59]:

$$\text{KHCO}_3 < \text{Water} < \text{Water mixed with Halon 1301 } (\text{CF}_3\text{Br}) < \text{Halon } (\text{CF}_3\text{Br})$$

Water is two to three times as effective as potassium bicarbonate, whereas the hybrids and Halon alone are only marginally superior to water. Triggered barriers have the following advantages over the conventional barriers:

probability that a few shelves will be at some optimum distance from the explosion origin to suppress the explosion flame. The distributed barriers were found to be effective in suppressing slow-moving to powerful explosions with static pressure of up to about 6 bar, dynamic pressures of up to 1.6 bar, and flame velocities between 30 and 500 m/s.

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(1) Displacement of loaded mine cars, strong steel supports, machinery or other heavy objects, and bending of steel beams, which may serve as the best guide for ascertaining the direction in which the explosion force has travelled through the mine workings;

(2) Charring or burning of materials sensitive to flame such as paper, fibres of torn timber, and pieces of clothing which may give, on minute examination in the laboratory, valuable information on the passage of the explosion flame;

(3) Formation of traces of coke crust or globules on roadway supports or projections as well as on sides and roof, which not only indicate with definiteness the passage of a coal-dust explosion flame but may also throw light on the intensity of the explosion;

(4) Formation of soot deposit occurring as a layer on horizontal surfaces or as a lacy network or stringers at the roof, which would indicate the passage of the coal-dust explosion flame;

(5) Nature of death or injury of the victims, whether by burning, violence,

All the evidence and data collected must be marked on a large-scale plan of the working. By careful analysis of the evidence, the most likely sequence of events and the most probable causes of the origin and seat and spread of the explosion may be inferred.

In investigations of complex accidents, a number of hazard analysis and accident investigation techniques have been developed involving the application of scientific and management principles such as the Scientific Method (Research Technique) Change Analysis, Gross Hazard Analysis, Failure Mode and Effect Analysis, and Fault Tree Analysis, any one of which may be used in making decisions.

Investigations into the causes and circumstances of a mine explosion are usually carried out by a Court of Enquiry set up by the State comprising, as its members, impartial persons who have had a wide experience of varied mining conditions and who are specialists in their field. The final report of the Court of Enquiry reveals in all probability the basic causes leading to the disaster and the roles of the mine management and mine workers.

The various precautionary measures and their combination that may be adopted to combat the explosion hazard in mines are:

(1) Selecting the proper type of explosives with low incendivity, i.e., with low explosion temperatures which will not adversely affect fragmentation;
(2) Selecting a drillhole pattern which will prevent rifling of the explosive energy from the drillhole [61];
(3) Accurate drilling of the holes for successful application of incendive explosives;
(4) Careful loading of the explosive charge in holes to maintain high compaction ratios at an effective level;
(5) Tamping inert stemming cartridges in the unloaded collar of each hole;
(6) Washing down the walls and backs in the development headings before blasting; and
(7) Creating water mist curtains at the face during a blast.

CHAPTER 3

Mine Gases

Normal ambient air is a mixture of gases containing 78.10 per cent nitrogen, 20.93 per cent oxygen, 0.03 per cent carbon dioxide, 0.94 per cent argon, and traces of hydrogen, helium and other rare gases together with a variable amount of water vapour (0.1 to 3 per cent, according to the relative humidity). 'Atmosphere' is another word for air.

Pure gases seldom exist in the workplace. Generally as air is circulated through a mine, its composition only changes a little. It loses some of its oxygen due to workers' breathing and oxidation processes in the mine. It picks up injury will result from repeated exposure day after day in workplace air. Determination of the threshold limit values is often a difficult problem. Every year the American Conference of Governmental Industrial Hygienists review and specify threshold limit values for a very large number of industrial materials, which are generally adopted as reference values throughout the world. Data available from extensive assessment studies should be taken into account in determining the proper values.

There are three types or categories of threshold limit values:

Threshold limit value – time-weighted average limit (TLV – TWA).

Threshold limit value – short-term exposure limit (TLV – STEL).

Threshold limit value – ceiling (TLV – C).

Time-weighted average (TLV – TWA)

detect and display the gas concentration of a single gas of concern or all gases simultaneously.

Gas concentration is commonly measured in percentage (%) by volume and parts per million (ppm). They are easily converted to one another by using the basic relation

$$\% \times 10000 = \text{ppm}$$

Contamination limits are only quoted in milligrams per cubicmetre (mg/m³). The following relation is used to convert concentrations from ppm at 20°C to mg/m³.

$$\text{mg/m}^3 = (\text{ppm} \times M) \div 24.4$$

where M is the molecular weight of the gas and 24.04 is a g as constant.

3.2 Oxygen (O_2)

can be revived by fresh air or oxygen. Increased oxygen demands during periods of hard work may result in oxygen-deficiency symptoms at much higher percentages. Oxygen deficiency in circulating air of self-contained breathing apparatus caused by the wearer of the apparatus forgetting to turn on the oxygen cylinder valve leads to similar effects. The threshold limit value of oxygen is given as 19.5 ppm.

The adequacy of oxygen in mine air may be ascertained by means of a flame safety lamp, the flame of which becomes dim and small and finally goes out when the oxygen content falls to about 16 per cent, whether carbon dioxide be present or not. Small amounts of methane tend to decrease this value somewhat; for example, when the atmosphere contains 1 per cent methane, extinction occurs with about 15 per cent oxygen. An acetylene lamp does not give a warning of the presence of an irrespirable atmosphere as it burns in oxygen concentrations up to 12 per cent and is extinguished at 10 per cent.

The inhalation of pure oxygen under normal conditions for a limited period of up to 6 hours has no harmful effects. With self-contained breathing apparatus,

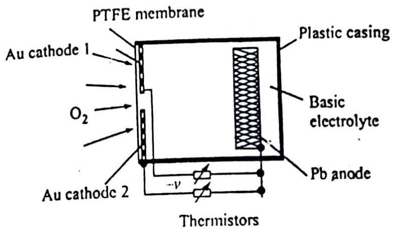
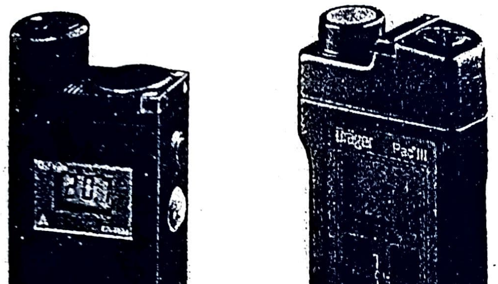


Figure 3.1 Fuel cell

The main advantages of the electrochemical sensing cell are its remarkable stability and rapid response over a long period and negligible influence of pressure or humidity variations. The microprocessor controls the measurement and processes the temperature-compensated signal to display a non-ambiguous value.

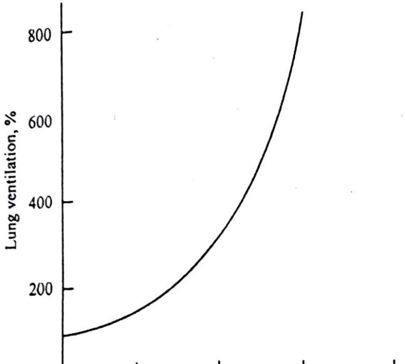


3.4 Carbon Dioxide (CO_2)

Carbon dioxide is a colourless, odourless gas having a slight acid taste. It is about one-and-a-half times heavier than air and, in consequence, is found in low-lying places in mines. It is extremely soluble in water, 100 volumes of water dissolving at 15°C, 100 volumes of the gas forming a weak acid. The critical temperature is –31.9°C. Carbon dioxide is not combustible and does not support combustion nor sustain life.

It is produced in mines from the following causes:

1. Breathing of men (average 50 litres per hour per worker); produced in the human body and exhaled in the respiratory process.
2. Burning of flames (0. 15 to 5 litres per minute per lamp);
3. Decay of mine timber;



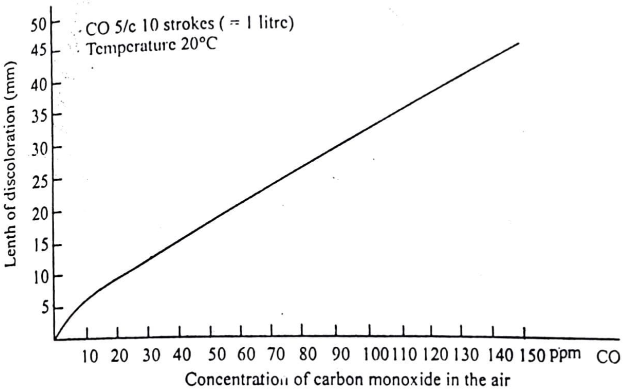


Figure 3.4 Calibration curve for the Dräger carbon monoxide 5/c tube

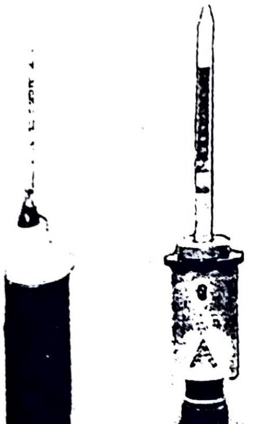


Figure 3.5(d) shows MSA/Auer Toximeter II which has a stroke volume (electronic stroke 100 + 5 ml) which is regulated by a temperature-compensated differential pressure sensor. The detector tube is inserted on the inlet side.

The Dräger Multigas Detector (Fig. 3.5(c)) has a hand-operated bellows pump which supplies 100 ml at each stroke. The pumphead has an aperture into which the detector tube to be used is inserted. The bellows, made from neoprene, has a valve which closes when the air sample is drawn in and opens on squeezing the bellows. To draw a sample, the bellows are first compressed and then released, the end of the suction process being reached when the limit chain is taut. The opening time during which the sample is drawn depends on the resistance of the detector tube. It may vary from 3 to 40 s. The range of the opening time is indicated in the instructions for use of the detector tubes.

The Dräger carbon dioxide detector tube has two measuring ranges of 0.1 to 1.2 Vol - % (five strokes) and 0.5 to 6 Vol.-%~ (one stroke). The tube contains a pre-cleansing layer and a white indicating layer of the chemical, hydrazine, which changes colour to bluish-violet due to reaction with carbon dioxide. The presence of other

The Auer ACO_2 hand-held indicator (Fig. 3.5(e)) uses an electrochemical sensor operating over the concentration range 0.03–100 Vol. -%. It samples the atmosphere by diffusion. An easily replaceable electrolyte chamber provides the sensor a life expectancy of up to 12–24 months, depending on its operation. The recommended replacement of electrolyte is every three months. The instrument is controlled by a microprocessor and powered by four standard 1.5 -V batteries. All functions can be selected with a single pushbutton. The instrument has two adjustable alarm thresholds, factory-set, at 0.5 Vol -% CO_2(TLV) and a main alarm at 1.0 Vol.-%. It is housed in dust- and water-proof antistatic material and weighs approximately 310 g with batteries.

3.5 Carbon Monoxide (CO)

Carbon monoxide is a colourless, tasteless, odourless, and non-irrit

haemoglobin depends on the exposure time, the CO-concentration, and the activity being performed. An individual's response will vary depending on his age and general physical condition. Carbon monoxide is non-toxic to insects and other lower forms of life which have no red blood cells.

The reaction between O_2, CO, and haemoglobin is reversible and may be represented as

$$HbO_2 + CO \rightleftharpoons HbCO + O_2$$

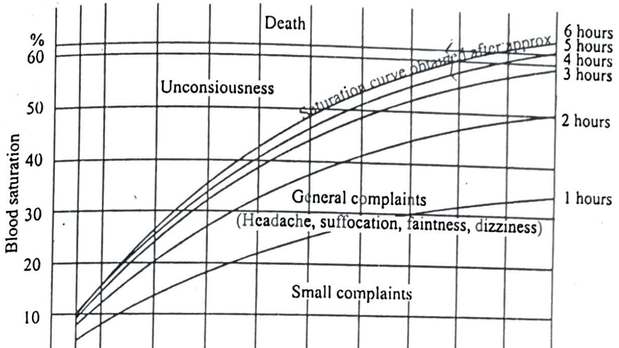
The equilibrium constant

$$K = \frac{[O_2].[HbCO]}{[CO].[HbO_2]}$$

of immediate death. Figure 3.6 gives a chart, 'Physiological effects of carbon monoxide on human beings,' prepared by the Staubinstitut für Staublungenforschung and Gewerbehygiene, Germany [65]. It shows the relationships between the CO-concentration (range 0.005 to 0.1 per cent CO), time of exposure, and percentage saturation of blood as well as the after-effects or symptoms of poisoning observed on a normal healthy human being. It is clear from the chart that even very small concentrations of the gas are harmful. From the chart, Table 3.1 is prepared to serve as a guide when working in mine atmospheres containing carbon monoxide. The threshold limit value is 50 p.p.m. A 70–80% blood saturation by CO will cause death.

Table 3.1 Physiological effects of mine air co-concentrations

CO concentration	Time of exposure	Symptoms	Work in such atmospheres



For these reasons, canaries as living detectors have been entirely replaced by colorimetric indicating detectors and electronic gas measuring instruments.

The colorimetric stain tube methods developed work on the same principle. In each case, the manufacturer develops a detector tube that changes colour along the length of the tube in proportion to the concentration of the gas in the air. A known quantity of air, usually in multiples of 100 ml, is drawn through the tube after the two glass ends have been broken and the tube is properly inserted into the pump. Most pumps are designed to draw 100 ml per stroke.

The MSA Universal Tester/Detector, Auer Gas Tester, and Dräger Multigas Detector using length-of-stain detector tubes, provide a rapid method of on-the-spot measurement of the concentration of carbon monoxide in the range 10 to 1000 ppm or 0 to 3000 ppm, depending on the amount of air sampled.

The MSA detector tube contains yellow silica gel impregnated with palladium sulphate and ammonium molybdate. When an ambient air sample containing

Auer developed the Toximeter II for use in conjunction with Auer detector tubes, which uses a battery-operated pump electronically controlled to draw a sample of air through the detecttor tube at a rate of 100 ml per basic time unit of 25 s. The basic time unit can be automatically multiplied by means of an electronic multiplier. The number of measurements that can be made per battery charge varies between 100 and 800, depending on the time of measurement. The chief advantages of the Toximeter compared with the manually-operated detectors are time saving through automatic operation, for both intermittent measurement and continuous air monitoring, and greater reliability. It can be used for collecting samples in or drawing samples from sampling tubes or sample containers and as a pump of a.smoke generator to generate smoke with smoke tubes or smoke cartridges.

The Dräger Quantimeter is an automatic model of the Dräger Multigas Detector based on bellows pump. A fixed volume of 100 ml is drawn through the detector

PPM CO
MiniCo™
Carbon Monoxide
Indicator
Model
II
Part No.

For continuous monitoring of the ambient air for CO-concentration, hand-held personal monitors have been developed by Auer (TOX-PEM CO), Draeger (Oxymeter/Toximeter PACIII), Ricken Keiki (Model CO-82), Sabre (Model PM 200), and others which are small and lightweight in design and can be carried in the breast pocket or worn around the neck.

The portable monitoring instruments using fuel cells, also called indicators, provide direct readings as well as audible and visual alarms if preset at 50 ppm concentration or at any other desired level. They are used for on-the-spot measurement of carbon monoxide in air for fixed-point continuous monitoring in any location in a range of 0 to 100 or 0 to 500 ppm. The rechargeable 2.4 V Ni-Cd battery pack sealed in a plastic case provides 8 to 10 hours of sampling in a fully charged condition. A small pump draws an air sample into the inlet through a flowmeter and a factory-set flow control valve. A portion of the sample is routed through an interferant filter to minimize their effects and into the sensor sample

In dusty environments, an external dust filter would be necessary. Use of a sample humidifier is recommended when sampling for more than one hour per day and/or the relative humidity is less than 50 per cent.

A variety of sampling lines and probe rods are available for testing atmospheres in remote locations. The instruments are equipped with a carrying handle and a shoulder–waist harness.

Figure 3.9 shows the block diagram of portable carbon monoxide indicators.

3.6 Hydrogen Sulphide or Sulphuretted Hydrogen (H_2S)

Hydrogen sulphide, referred to as stinkdamp in the early days of mining, is a colourless gas having a sickly sweet taste and a disagreeable odour of rotten

3.6.1 Detection and estimation

Hydrogen sulphide can be detected by its smell of rotten eggs even at concentrations as low as 0.0001 per cent (1 ppm). This distinctive odour, however, is noticed only in concentrations less than 0.2 per cent as at higher concentrations or following continued exposure to concentrations as low as 0.01 per cent, the gas paralyzes the sense of smell and the sensory response is not proportional to the concentration of the gas.

Hydrogen sulphide can be detected by using colorimetric indicating detectors. The MSA detector consists essentially of palladium chloride in an acetone–water solution, sealed in a small glass tube which is surrounded with absorbent cotton. The presence of hydrogen sulphide is indicated by a colour change to dark in the cotton when the tube is crushed between the thumb and forefinger; the discoloration is compared with the scale on a standard colour chart supplied.

Length-of-stain hydrogen sulphide detectors using detector tubes are also exceed preset limits on the low and high ranges. The measuring ranges are 0 to 20 and 0 to 100 ppm H_2S in air with a minimum detectable concentration of 0.5 ppm. A small pump draws a sample of air which is analyzed for hydrogen sulphide. The instrument can be carried using the handle or slung over the shoulder using the strap. Such indicators are manufactured by leading manufacturers of rescue and gas detection equipment.

3.7 Sulphur Dioxide (SO_2)

Sulphur dioxide, the primary oxide of sulphur, is a colourless, suffocating, and irritating gas with a strong pungent sulphurous smell. It is 2.21 times heavier than air and it tends to settle into low-lying areas.

Sulphur dioxide is produced with other gases in mines mining sulphide ore from blasting, mine fire and explosions, and as a product of complete combustion of fuel in diesel engines. As the production of sulphur dioxide in the exhaust

colour and have a smell of blasting powder fumes. They are easily dissolved by the moisture in mine air.

Nitrous fumes are formed during blasting by high explosives such as gelatine dynamite due to incomplete detonation. They tend to settle in the freshly blasted and dry muck pile, being about one-and-a-half times heavier than air. They are released when the muck pile is disturbed, posing a hazard to the loader-operator, or anyone else in the vicinity.

Nitrous fumes are also produced during the combustion process inside the cylinders of a diesel engine. The amount formed depends on the maximum temperature reached in the cylinders during the combustion cycle, the amount of oxygen available for completion of the chemical reaction, and engine maintenance. The principal oxide of nitrogen formed is nitric oxide which however, being unstable, rapidly oxidizes to nitrogen dioxide in the presence of oxygen, both in the exhaust pipe and in the mine air.

Nitrous fumes are highly poisonous. They are 7 to 10 times more toxic than unburnt hydrocarbons to harmless CO_2 and water while rhodium reduces the harmful NO_x to harmless nitrogen.

3.8.1 Detection and estimation

Nitrous fumes can be detected by colour-stain detector tubes in concentrations of up to 100 ppm. The Dräger detector tubes measure nitrous fumes ($NO + NO_2$) in the ranges 0.5 to 10, 2 to 50, and 5 to 100 ppm. The detector tube contains a reddish-brown oxidation layer containing a chromium compound and a yellow indicating layer of N-diphenyl benzidine which changes colour to dark bluish-grey in the presence of the fumes. The oxidation layer oxidizes NO to NO_2.

Personal nitrogen oxide samplers and hand-held direct-reading NO_2 detectors are now available in the market.

3.9 Hydrogen (H_2)

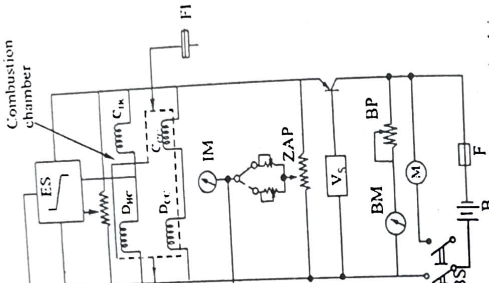
The presence of methane in a mine causes such a mine to be called a 'fiery', mine. Methane chiefly occurs in virgin coal seams (including anthracite) and associated strata as free and adsorbed gas. It was formed as a by-product during the so called coalification process of vegetable matter when the seams were formed. Bituminous coals contain larger amounts of the gas than other coals. Methane enters mine workings out of coal seams and their adjacent strata as well as out of coal being mined. It may also be found in the region of faults, dykes, slips, or rolls in roof cavities, above falls, high places and abandoned workings. It is given off in the form of (i) regular, slow and continuous emissions, (ii) blowers, and (iii) gas outbursts. Methane emission rates are often unpredictable because of the varying interactions between geology, mining, ventilation and methane control measures/systems adopted owing to site-specific conditions and mining practices. Methane has also been reported to have occurred in salt, potash, sulphur, lead-zinc, silver, gold, tin and diamond mines but is much less common.

Methane is also found in the surface atmosphere in coalfields, being released during mining of coal seams and liberated into the atmosphere through the mine heated filament (active sensor) forms part of the Wheatstone bridge circuit, the change creates an imbalance in the circuit, which results in a deflection of a meter pointer in proportion to the methane concentration. The ability of a gas to conduct heat (the coefficient of thermal conductivity) is a specific physical property of the gas. Table 3.2 gives the relative heat conductivity of some gases taking the heat conductivity of air as 1[67].

Table 3.2 Relative heat conductivity of some gases

Air	1.00
Oxygen	1.01
Nitrogen	0.99
Hydrogen	7.13
Carbon dioxide	0.58
Carbon monoxide	0.96

(d) Improved detector filament having a long life;
(e) Simplicity of operation by means of push button(s);
(f) Digital readout;
(g) Recharging of battery without having to open the instrument case, with automatic switch-off; long battery life;
(h) Robust construction with maximum shock and vibration resistance and easily accessible interior;
(i) Telescopic sampling probes for sampling in inaccessible places;
(j) Autocalibration,
(k) Continuous operation when required; able to store large numbers of gas readings
(l) An adjustable alarm (audible and visual)
(m) Meter illumination in dark places;
(n) Wide temperature range;
(o) Sensor immune to noise; and



talytic combustion–heat conductivity
nciple

heat conductivity, and catalytic combustion–
cating meter; CP: calibration potentiometer;
ry meter; BP: battery check potentiometer;
electronic switch.

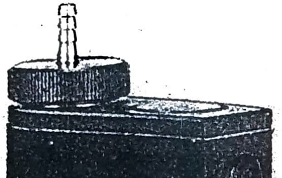
Figure 3.11 gives schematic diagrams of methane detectors/ methanometers based on the principles of catalytic combustion, heat conductivity, and catalytic combustion–heat conductivity.

Spot measuring methanometers are also designed incorporating a warning device for use by special patrols or investigations.

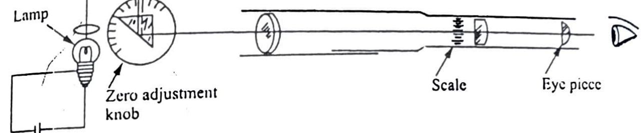
For continuous measurement

Methanometers for continuous determination of methane levels at strategic locations near the face and on equipment in mine workings are called methane monitors or automatic methanometers. They provide visual and audible alarms and, in some instruments, a continuous record of the methane concentration. When provided with an external relay unit, they can be used to automatically switch off electrical machinery as required. They are operated continuously from mine power mains or from a rechargeable nickel–cadmium battery (12 V, by an absorption tube containing silica gel and soda-lime; dust and dirt are removed by an easily replaceable mechanical filter.

The detector and compensator sensors form part of a balanced electrical bridge circuit. When a mixture of methane in air is drawn through the instrument, it comes into contact with the detector sensor, the latter will cool due to heat loss at a rate depending on the concentration of methane. Thus, the voltage across the sensor will decrease, thereby upsetting the bridge circuit and resulting in deflection of the meter pointer.



A well-known instrument approved for use in mines is the Auer Methanometer M_5 10B, which is calibrated for a methane range of 5 to 100 Vol. per cent CH_4 with an accuracy of indication of ±0.2 Vol. per cent CH_4. A single charge


Plane parallel glass plate
Fresh air chamber
Prism
Gas chamber
Fresh air chamber
Lamp
Zero adjustment knob
Scale
Eye piece

If z is the number of fringes shifted, $(n_m - n_a)l = z\lambda$.

The amount of fringe shift is, therefore, proportional to $(n_m - n_a)$, which means that the larger the value of n_m, the more sensitive is the instrument.

If the concentration of the gas in air is x vol. per cent, the refractive index of the mixture of gas and air is given by

$$n_m = \left(1 - \frac{x}{100}\right) n_a + \frac{x}{100} n_g$$

where n_g is the refractive index of the gas.

To take readings, the gas chamber is cleansed with fresh air by allowing it to flow through it by squeezing the aspirator bulb about five times. The press button switch is then operated to illuminate the occular scale. Interference fringes will be observed through the eyepiece. The fringes consist of two black lines or stripes with coloured ones on either side of them. Of the two black lines, the one on the right is selected as the 'index stripe'. The zero adjusting knob is next rotated until this black stripe shifts and coincides with the zero line of the ocular scale. This operation is called zero-setting of the instrument and the instrument is now ready for use (Fig. 3.15). Next, the aspirator bulb is squeezed at least five times to draw the air to be sampled into the gas chamber. After pressing the switch, the new position of the index stripe on the ocular scale directly gives the percentage of methane in the air sampled.

The measuring ranges are 0 to 6 per cent; 0 to 10 per cent; and 0 to 100 per

Care and use of methanometers

There is a wide choice in manometers and most methanometers designed today require relatively less maintenance and are easy to repair and use. Although designed sufficiently rugged for mining use, being precision instruments, all electronic gas detectors must be carefully and regularly maintained and handled with reasonable care and intelligence at all times so that mine safety is not jeopardized.

The routine inspections, maintenance and testing of hand-held methanometers are usually carried out in the lamproom by competent persons who must check the following with the fully-charged battery before the start of every shift:

(a) the external condition of the instrument;
(b) the proper assembly of the instrument case and the indicating meter window for any damage;
(c) battery voltage; and
(e) zero and calibration checks

alpha particles, beta particles, and gamma rays are emitted. Thus, airborne radiation is a mixture of radon and its daughters. On account of the short half-lives of radon daughter products, their concentrations rapidly increase when a source of radon is present but rapidly decay when they are separated from the radon source, resulting in a significant variation with location and time as the ventilating air flows through the mine workings. Thus, the operation of any uranium mine cannot be conducted without extensive impact on the environment throughout the life of the mine.

Table 3.3 Decay series of radon to lead

Name	Atomic No.	Mass No.	Range in air (cm)	Half-life	Principal radiation
Radium-226 (^{226}Ra)	88	226	3.39	1600 years	Alpha

Radon is found in trace quantities throughout the world in almost any kind of soil and rock in the earth's crust. Granite rocks contain more uranium, thorium, and potassium than any other kind of rock. Radon originates from man-made sources like nuclear power production and uranium mining. The United Nations Scientific Committee on the effects of Atomic Radiation (1988) has estimated that exposure to natural sources contributes more than 98% of the radiation dose to the population (excluding medical exposure). The global average human exposure from natural sources is 2.4 mSv/year (millisieverts per year), the variation depending on several factors such as height above sea level, the amount and type of radionuclides present in the soil, and the amount taken into the body through air, food, and drinking water.

Radon occurring in small concentrations in most hard rock mines may not pose any significant occupational hazard. Coal seams are also known to contain

The practical unit of exposure to randon daughters generally adopted is the working level (WL) which is defined as that concentration of daughter products in one litre of air which, on ultimate decay, liberates a total energy equal to the emission of 1.3×10^5 MeV (mega electron volts) of alpha energy. In SI units, the WL corresponds to 2.1×10^5 J/m^3.

As air passes through the mine workings, the working level of the air increases.

In mining environments, a special unit, the working level month (WLM) is used to measure workers' exposure resulting from inhalation of air containing 1 WL of radon progeny for one working month. Exposure to 1 WL for 170 hours gives a dose of 1 WLM.

The International Commission on Radiological Protection recommends that all exposures (doses) be kept as low as is reasonably achievable (ALARA principle) which will ensure that mine operators do more than just meet the regulatory standards.

while the standard for the public dose limit is 1 mSv/y ear above the natural radiation background level, allowing a higher dose in any one year.

The ICRP recommendations have been accepted internationally by the International Atomic Energy Agency (IAEA), the World Health Organization (WHO), and the International Labour Organization (ILO) and are followed all over the world. In India, the Atomic Energy Regulatory Board (AERB) is the national competent regulatory authority which has accepted the ICRP standards and is primarily responsible for radiation protection and monitoring in uranium mining, milling and ancillary operations, and nuclear power stations.

The relationship between the dose in WLM and the actual dose in sieverts to the target tissues in the lung is extremely complex, being dependent on many factors. The ICRP 66 (1994) suggests a conversion factor of 1 WLM = 5 mSv for occupational exposures, while some other studies recommend 10 mSv/WLM.

For monitoring radiation protection for ensuring compliance with regulation requirements, two types of monitoring programmes must be undertaken: a survey

Other sources of randon on the mine surface as a consequence of mining uranium ore are ore stockiles, mine waste rock piles, ore transportation, mine water disposal ponds, mill tailings ponds, crushing areas in the mill, and yellow cake packaging areas. Techniques have been developed to control the radioactive emanations from solid and liquid wastes produced to minimize any damage which may be caused to the environment.

Although one might expect radon daughters to travel over long distances on the earth's surface, in practice, experience has shown that the radiation levels are very low due to the combination of rapidly increasing area with distance and vertical mixing with air due to air turbulence.

Although the general observation usually made is that lung cancer deaths are higher than in the general population, it got disproved by French uranium miners who after working over a period of many years did not suffer from lung cancer any more than the general population in France [69]. This was made possible by enforcement of stringent control measures for protection of the environment to be asphyxiating in character, although it may be mixed with gases such as carbon monoxide and other vapours in proportions too dangerous to inhale.

3.13 Gas Analyzers

These are extremely accurate instruments used for analysis of air samples containing a mixture of gases in the laboratory or for continuous ambient air monitoring in mines.

Infrared analyzers

The Luft-type infrared analyzer provides an accurate and continuous analysis of a single selected gas in a mixture of gases or vapours in real time. It operates on the non-dispersive (NDIR) principle based on the absorption of infrared

where I is the transmitted intensity;
I_0 is the incident intensity;
k is a dimensionless extinction coefficient;
d is the length of path;
c is the gas concentration.

As the extinction coefficient for a given gas is a constant quantity, the transmitted radiation after absorption is a function of c and d. The length of the path in gas must, therefore, be selected depending on the expected gas concentrations to be measured. The higher the concentration, the shorter is the length of path.

Figure 3.18 gives a schematic diagram of an infrared analyzer. The instrument comprises an infrared source, a rotating interrupter or chopper vane, two side-by-side vertical cells or tubes, two receiving cells, a detector or diaphragm cell are also available in stainless steel, nickel, and monel metal. The internal surfaces are highly polished to give the best optical and chemical conditions for analysis. The ends of the tubes are sealed with windows of material (sapphire, quartz, calcium fluoride, barium fluoride, etc.) transparent to the radiation wavelengths used.

The two receiving cells are an analysis receiving cell and a reference receiving cell which are placed beneath the two vertical tubes so that two beams of unequal energies passing out of the tubes during a measurement enter their respective receiving cells. They are filled with equal volumes of the pure sample of the gas to be measured so that the gas exerts an equal pressure on the diaphragm.

During a measurement, the amount of energy received by the analysis receiving cell remaining after absorption in the sample tube generates heat in the cell, which is related to the concentration of the gas to be measured in the mixture.

degree of affinity of the molecules to the stationary phase, the molecules of the various components have different rates of progression along the column and reach the end of the column at different times thereby resulting in the separation of the various components of the mixture into discrete bands that can be analyzed qualitatively and quantitatively as they are swept by the carrier gas to a detector. A record of the effluent elute from the column after injection plotted against time is called a chromatogram. Figure 3.19 shows examples of good and bad separation of two components and Fig.3.20 shows a typical chromatogram of a single compound. Chromatography can be divided into two categories: gas - liquid chromatography and gas-solid chromatography. Gas-liquid chromatography is mostly used for environmental analysis of liquid and gaseous mixtures. It is a

for analysis of samples of mine atmosphere in the must be adjusted to provide optimum column efficiency which can be ascertained by observing the peaks plotted on the strip chart of the recorder. As a general rule, low flow-rates and low temperatures will result in broad peaks which yield unsatisfactory quantitative data. Larger flow-rates will yield narrow width of peaks thus providing better resolution of adjacent peaks. As a starting point for packed columns, a flow-rate of 60 ml per minute is recommended.

Two designs of columns are used in gas chromatographs: packed columns and capillary columns.

Packed columns are typically a glass or stainless steel coil containing finely divided inert solid support material (e.g., diatomaceous earth) coated with a liquid or solid stationary phase. The nature of the coating material will be determined by the sample to be analyzed. Reliable information on their selection is available from the manufacturer. Packed columns have inner diameter of 2 to 5 mm and a length of 1.5 to 10 m.

Capillary columns are tubular columns having a very small internal diameter. Most columns are made of thin fused silica with a polymer outer coating and an inner coating of an active organic compound as stationary phase material. The

molecules producing ions. The ions are attracted to the collector electrode producing electrons and the resulting electron current signal is amplified.

A FID is extremely sensitive to all organic compounds including hydrocarbons with a large dynamic range but is not sensitive to many inorganic compounds. Its only disadvantage is that it destroys the sample. It is not sensitive to CO, CO_2, H_2O, N_2 or O_2 which are normal flame products. Using Porapak-N as column material, the FID is used for measurement of CH_4, C_2H_6, C_2H_4 in mine atmospheres.

A TCD monitor changes in the thermal conductivity of the effluent. It can detect any sample molecules (organic or inorganic) that have thermal conductivity different from that of the carrier gas. It uses an electrically heated wire or thermistor as the sensing element whose temperature depends on the thermal conductivity of the effluent flowing around it. Changes in thermal conductivity causes a temperature rise in the element which is sensed as a change in resistance. The TCD consists of four tungsten–rhenium filaments in Wheatstone bridge configuration.

The TCD is commonly used for analysis of O_2, N_2, CO, CO_2, H_2S, NO, NO_2

3.14 Mine Environmental Monitoring and Control

The effectiveness of engineering control measures for compliance with mining regulations can be ensured only if methane concentrations in mine working are acurately monitored.

The introduction of continuous remote environmental monitoring and control system in mines, particularly in modern coal mines, is a great step forward in achieving increased mine safety, higher productivity, and effective utilisation of manpower. It provides, through alarms and displays, early warnings of impending abnormal and dangerous conditions or situations in mine workings such as spontaneous combustion and sudden emissions or layering of firedamp so that timely appropriate corrective action may be taken. Knowledge of average methane emission during the extraction of a seam as well as of the maximum actual-to-average emission will enable better design of mine ventilation and methane drainage systems.

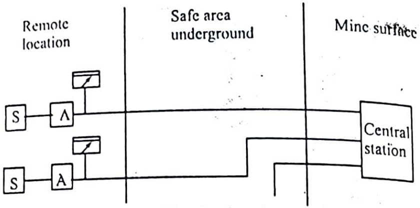
The environmental monitoring and control system provides measuring facilities to monitor and control

3.14.1 Environmental monitoring systems

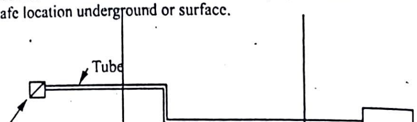
Depending on the number of underground sample points and the number of gases to be measured, one or more of the following basic systems of atmospheric monitoring may be used (Fig. 3.22):

(1) Where only a relatively small number of underground monitoring points are involved, sensors or self-contained analyzers are used and the output analogue signals, after amplification, are transmitted by a direct multi-core cable through an isolating zener safety barrier to a surface-mounted central monitoring station with data recording and audible or visual alarms systems. The system is simple and relatively inexpensive to install.

(2) The 'tube-bundle system' with individual nylon/polythelene sampling tubes going to the sample points to aspirate gas samples for analysis by a microprocessor-based analyzer (or analyzers) at a central station on the surface, which measures or derives the various parameters. The parameters that can be handled by the system are methane, oxygen, nitrogen, carbon dioxide, carbon monoxide, Graham ratio, and oxygen deficiency.



(i) System with sensors, amplifiers, local read-out (optional) at remote location in a hazardous area. Display, alarms at safe location underground or surface.



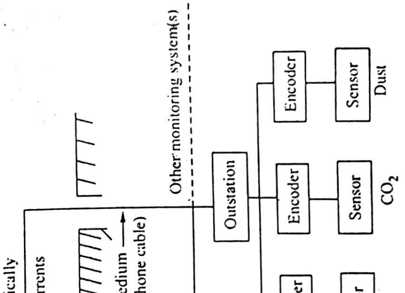
(2) Accuracy of equipment;
(3) Interference by other gases or electrical noise;
(4) In-built safety features – intrinsic safety;
(5) Ability to operate even when mine power is cut off for any reason;
(6) Ease of maintenance, availability of supplier technical back-up;
(7) Simplicity and sturdiness of construction, flameproofness and vandal-proofness;
(8) Speed of response;
(9) Accessibility for alarm-setting by underground personnel;
(10) High degree of reliability to provide necessary protection for which they are made;
(11) Real-time picture; and
(12) A sense of responsibility on the part of the mine management and technology gap at the mine.

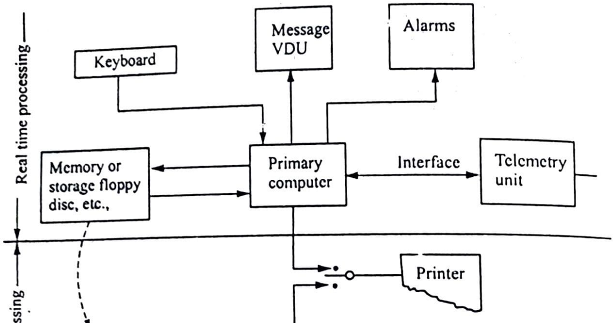
3.14.2 Mine monitoring and control system

A mine monitoring and control system utilizes multiplexing, data processing, and microprocessor techniques. The connected computers and their software allow presentation of environmental information, alarm conditions, data storage, and calculations.

A mine monitoring and control system (Fig. 3.24) may be considered to be made up of the following basic parts:

(1) A central station or master station located at the mine surface in a control room or a mine control centre comprising a central processor or telemetry unit that controls the operation of the transmission system and collects,


Other monitoring system(s)
Outstation
Encoder
Encoder
Sensor
Sensor
CO$_2$
Dust
hone cable)

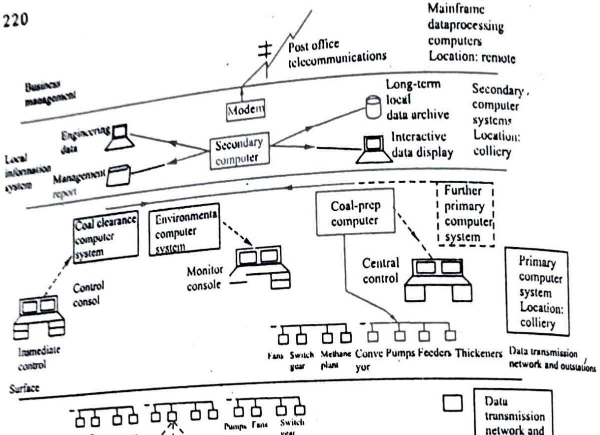

Real time processing
Keyboard
Message VDU
Alarms
Memory or storage floppy disc, etc.,
Primary computer
Interface
Telemetry unit
Printer
ssing

set of computer equipment, a basic computer programme called an 'operating system', and a set of application programmes which may be independent of the data transmission system to monitor and control a specific plant. The principal features of the MINOS system are:

(1) Use of standardized primary and secondary computers as well as software;
(2) Same keyboard;
(3) Similar displays on the VDUs irrespective of the manufacturer; and
(4) Built-in flexibility which permits operation of the system to suit local circumstances.

The MINOS became a standard monitoring and control system for British collieries.

Recent advances in technology such as microcomputers and touch-pad keyboards have been included in the concept Micro-Minos system which has


Post office
telecommunications
Mainframe
dataprocessing
computers
Location: remote
Business
management
Modem
Long-term
local
data archive
Secondary
computer
systems
Location:
colliery
Engineering
data
Secondary
computer
Interactive
data display
Local
information
system
Management
report
Coal clearance
computer
system
Environmenta
computer
system
Coal-prep
computer
Further
primary
computer
system
Monitor
console
Central
control
Primary
computer
system
Location:
colliery
Control
consol
Immediate
control
Fans
Switch
gear
Methane
plant
Conve
yor
Pumps
Feeders
Thickeners
Data transmission
network and outstations
Surface
Pumps
Fans
Switch
gear
Data
transmission
network and

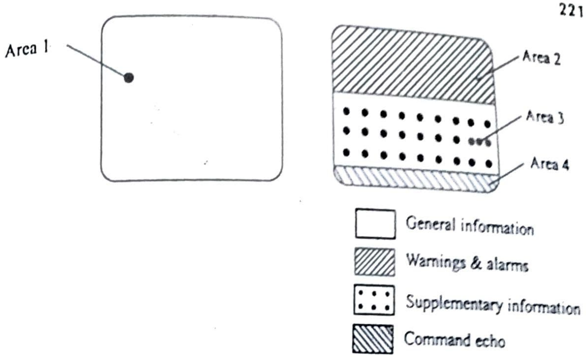

Area 1
Area 2
Area 3
Area 4
General information
Warnings & alarms
Supplementary information
Command echo

Figure 3.28 √DU screen allocations.

In the Micro-Minos system of HSDE Ltd., the data are automatically stored

(1) Analogue transmission systems;
(2) Digital transmission systems
(3) Frequency-based communication systems

Transducer outputs are electrical signals which may be analogue – current voltage – or binary numbers.

The analogue transmission system is the simplest and most widely used technique in the general engineering industry. In mine applications, 0 to 10 V, 0.4 to 2 V DC and 0 to 1 mA and 4 to 20 mA signals are commonly used.

Owing to some inherent disadvantages in transmission of analogue voltage or current signals, digital and frequency-modulated signals are widely used in mines employing analogue voltage signals.

signals are in the form of ON/OFF signals generated by the transducers

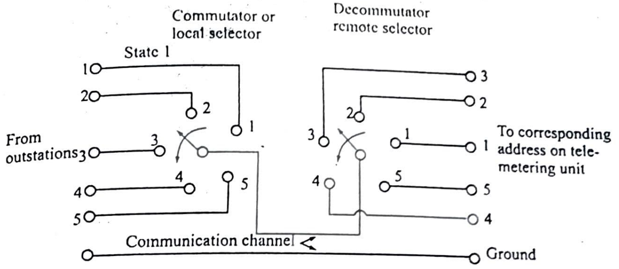

Commutator or local selector
Decommutator remote selector
State 1
1
2
3
4
5
From outstations
2
1
3
4
5
3
2
1
4
5
3
2
1
5
4
To corresponding address on tele-metering unit
Communication channel
Ground

Figure 3.29 Analogue time-division multiplexing system (TDM).

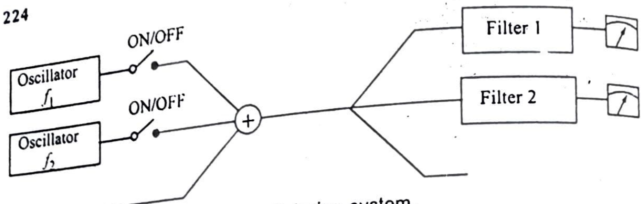


Figure 3.31 Frequency-division multiplexing system.

FDM are its simplicity and insensitiveness to electrical disturbances. The disadvantages, however, are that the system is expensive, the number of channels which may be used is restricted (maximum 40), the accuracy of the transmitted data is restricted, the second and the third harmonics can cause false detection and, hence it is unreliable compared with the digital system.

computer or control console, which can then be directly displayed. The outstation design contains local display of status.

3.14.2.5 Transducers

Transducers (also called sensors, monitors, detectors) are used to measure the variable parameters and are sited as near as possible to the sampling points. They produce information in the form of an electrical signal either as an analogue voltage in the range 0.4 to 2 V, 4 to 20 mA or a set of voltage-free microswitch or relay contacts at one or more preset alarm levels.

A transducer consists of a sensing element which may or may not be designed integral with an electronic circuit which will convert the signal received from it into an electric analogue voltage for further processing for transmission. The power supply is usually a battery. Most transducers use liquid crystal display (LCD) for local indication and calibration or a light emitting diode (LED). Figure 3.32 shows the transducer system. When selecting a transducer, the cost, accuracy, life, selectivity, response time, reliability, maintainability, and power requirement must be taken into account. The

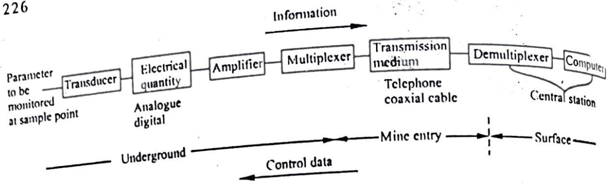

Information
Parameter to be monitored at sample point
Transducer
Electrical quantity
Analogue digital
Amplifier
Multiplexer
Transmission medium
Telephone coaxial cable
Demultiplexer
Computer
Central station
Underground
Mine entry
Surface
Control data

Figure 3.32 Transducer system.

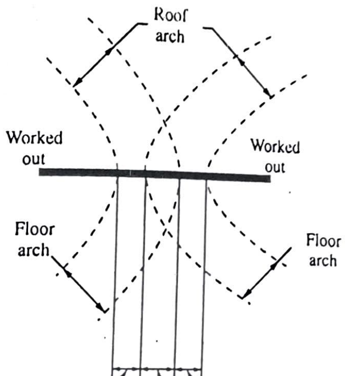
CHAPTER 4

Outbursts of Coal and Gas

An outburst is a phenomenon characterized by sudden displacement or ejection into mine workings of either large volumes of fragmented rock with or without gas or large volumes of gas alone. The phenomenon is generally attributed to high gas pressures and/or overstressing due to accumulated stresses in rocks, although its mechanism is still not fully understood.

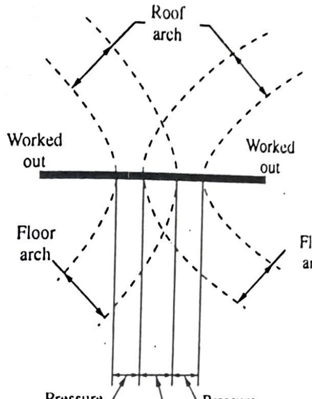
coal and the quickness of the desorption ability. In large outbursts, both gas outbursts and bumps may, however, occur at the same time due to joint action of rock and gas pressures. Several hundred tonnes of usually fine coal may be ejected in a fraction of a second and several hundred thousand cubic metres of gas may be evolved in large outbursts. The pressure of the enormous quantity of the instantaneously released gas may be sufficient to push back the ventilating intake air current and destroy the ventilation system. The gas emission generally stops rapidly but sometimes it may continue for a long period. Whereas CH_4 in the strata is a product of the coal-forming process, CO_2, with carbon dioxide outbursts, is introduced into the seams through igneous intrusions which cause partial devolatilisation of coal adjoining the intrusion with local increase in rank. Owing to the higher absorption capacity of coal for CO_2 than for CH_4, CO_2-outbursts are characterized by greater violence and seams containing CO_2 show the greatest proneness to outbursts.

Unlike bumps which often occur away from active faces, gas outburst



coal and the quickness of the desorption ability. In large outbursts, both gas outbursts and bumps may, however, occur at the same time due to joint action of rock and gas pressures. Several hundred tonnes of usually fine coal may be ejected in a fraction of a second and several hundred thousand cubic metres of gas may be evolved in large outbursts. The pressure of the enormous quantity of the instantaneously released gas may be sufficient to push back the ventilating intake air current and destroy the ventilation system. The gas emission generally stops rapidly but sometimes it may continue for a long period. Whereas CH_4 in the strata is a product of the coal-forming process, CO_2, with carbon dioxide outbursts, is introduced into the seams through igneous intrusions which cause partial devolatilisation of coal adjoining the intrusion with local increase in rank. Owing to the higher absorption capacity of coal for CO_2 than for CH_4, CO_2-outbursts are characterized by greater violence and seams containing CO_2 show the greatest proneness to outbursts.

Unlike bumps which often occur away from active faces, gas outbursts occur



An outburst in the face may occur when its front abutment pressure intersects the existing pressure zone of a worked-out face in an overlying seam. In semi-steep and steep measures, the coal tends to dislodge or run out aided by gravity (Fig. 4.3).

(11) Presence of a strong band or bands followed by a permeable and/or porous stratum and a coal seam or carbonaceous bed in the floor of the seam being worked, which is liable to floor outbursts [73].

(12) As the load stresses increase with depth due to superincumbent and lateral confinement and as the gassiness of the strata also increases with depth, outburst proneness increases with depth although it has not been categorically established in some coalfields. The shallowest outburst occurred at Collensville Colliery in Australia at 180 m depth. There is, however, no direct relationship between depth and proneness although the possibility of the incidence of higher at greater depths, exceeding about 1200 m.

(15) At points of converging pillar lines in bord and pillar workings.

(16) When working thick seams which cannot be mined to full thickness in one pass and are developed in the floor, leaving roof coal.

(17) When working seams of high-rank coals which are generally more prone to outbursts than lower-rankgcoals. The coals usually affected are those of relatively high-rank bituminous and anthracite.

(18) When practising destress blasting, also called 'shock blasting' or inducer shotfiring, which may precipitate on outburst immediately following blasting or after a delay of several hours.

(19) High rates of face advance which will increase the gassiness of face coal and increase proneness.

(20) When mining under a hilly terrain with crests and valleys.

Prevention of Outbursts of Coal and G

To reduce the incidence of outburst in a highly stressed area, the method has been modified by advance boring of one or more large-diameter holes (150 to 300 mm) which serve to degass and destress the coal face before the face is advanced.

Special precautions to be observed with inducer shotfiring;

(i) The number of shotholes in a round and charge per hole should be adequate.
(ii) Temporary and permanent supports should be set as soon as possible after firing and as close as possible to the heading face.
(iii) Electric power should be switched off prior to firing.
(iv) No person should be on the return side of the place and the persons engaged in firing shots should be at least 200 m from that place.
(v) There should be a ready means of communication between the firing station and the surface.
(vi) To reduce the damage from an ensuing outburst, a barricade may be

(5) A single-face advance is preferable to a two-face advance in two opposite directions, with the face making an angle of 10° to 15° with the direction of the main cleat of coal.
(6) In inclined seams, advancing the face in the dip direction is safest.
(7) With longwall advancing, the gate roads should either be very short or extend beyond the highly stressed front abutment zone.
(8) Caving or hydraulic filling/stowing should be practised, particularly in deep mines, as a method of ground control.
(9) Working a 'protective seam' underlying or overlying the suspect or protected seam with concentrated stress before mining the latter. The area affected by the vertical strata destressing lies between 50 and 100 m in both directions. If the protective and protected seams are worked at the same time, then the workings in the protective seam must be much ahead of those in the protected seam, by a distance which is determined by the vertical distance between the seams and the caving angle of the overlying strata. In very thick seams, the topmost slice, if extracted without leaving

(1) Underground observations
(2) Gas measurements
(3) Measurements on coal
(4) Microseismic measurement

4.2.3.1 Underground observations

Underground observations consist of observation of local audible and visible premonitory signs and use of simple instruments and methods. The warning signs usually given out in outburst zones several days or hours before their occurrence include a change in the seam thickness, cracking noise from the coal mass, higher resistance to drilling in coal, closure of shotholes during their charging, weighting of supports and occurrence of microfractures and break

4.2.3.3 Measurements on coal

Drillcuttings swell factor, given by the ratio

$$\frac{\text{Volume of drillcuttings}}{\text{Volume of hole drilled}}$$

is commonly used as a predictive index. It is important that the drill bit is sharp, the cuttings are collected in the high-pressure zone, and a standardized hole diameter is used. The volume of drillcuttings per metre length of drillhole is also used as a predictive index when it exceeds a certain value determined by experience with burst-prone areas.

Softening of coal due to increased fissuration takes place when approaching an outburst zone. This is revealed by a study of the microstructure of coal

In any comprehensive programme of research on outbursts of coal and gas, the following items of study/investigation in the laboratory, planning office, and field must receive attention:

(1) Gas-coal; confining rock permeability;
(2) Gas-coal adsorption and fast desorption characteristics;
(3) Seam gas compositions;
(4) Seam gas pressures;
(5) Face gassiness indices;
(6) Seam gas advance drainage techniques from underground and from the surface;
(7) Direction of coal cleating;

CHAPTER 5

Sampling of Mine Atmospheres and Interpretation of their Analyses

average composition. It is seldom necessary to take continuous samples as a number of grab samples taken at predetermined intervals dictated by the urgency of information will provide more information than one continuous sample. The introduction of computerized continuous mine environmental monitoring and control systems in mines, however, offers many advantages since they can provide round-the-clock information on air quality and quantity at sampling points and can give early warnings of developing abnormal situations.

5.1.2 Sampling in accessible places

When detecting spontaneous combustion in a suspected area, samples may be collected in the return airway of the area or in the supposed course of gases coming from the area before the gases get diluted. When sampling in a return airway, the sampling point or place should be selected not nearer than 20 m of of contamination of the moving air by the gases coming

samples through them to eliminate dilution or contamination of the samples. The samples should be marked whether the borehole was 'blowing', 'sucking' or relatively 'still'.

5.1.4 Construction of a sampling pipe or line at a stopping

When a stopping is constructed, unless conditions are unusual, a 25- or 50-mm diameter steel pipe should be built into it near the centre extending as far behind the stopping (greater than 20 m) as is convenient to reach but not too close to the fire, which may show too low or too high a concentration of fire products. Samples are drawn either through the pipe or through a 6-mm inside diameter seamless copper sampling line or tube inserted and properly fitted into the pipe so that the whole assembly is airtight (Fig. 5. 1). Plastic probes may also be passed through the pipe to take samples.

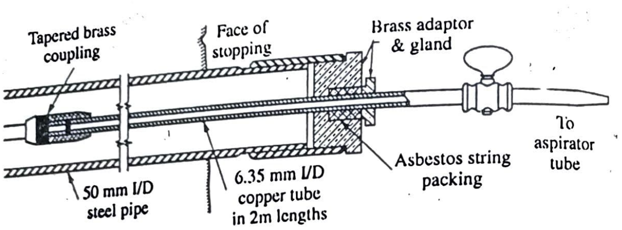
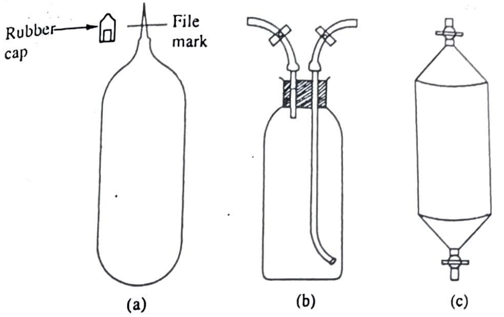
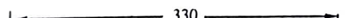
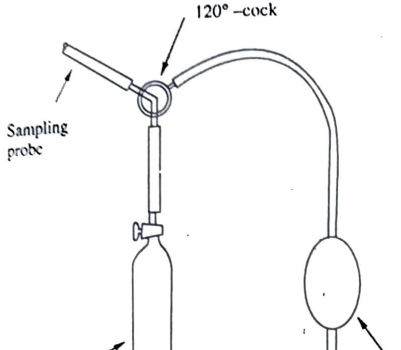

Tapered brass coupling
Face of stopping
Brass adaptor & gland
To aspirator tube
Asbestos string packing
6.35 mm I/D copper tube in 2m lengths
50 mm I/D steel pipe

Figure 5.1 Copper sampling line

water displacement, one of the short pieces of glass tubing joined at each end of the container has a capillary bore through it. The diameter of the capillary drains out in about 40 seconds [76].


Rubber cap
File mark
(a)
(b)
(c)





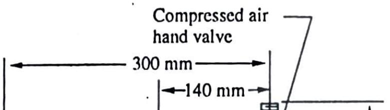
the cross-section by 10 strokes of a piston and the sample collected at 10 points using two strokes at each point (Fig. 5.4(b)).

5.1.6.3 Samples through sampling pipes of sealed-off areas

When a sampling pipe is blowing out, sample containers may be directly filled by gas displacement without the aid of an aspirating device. It is necessary for such direct sampling that the gas blows out of the sampling line over a longer duration with sufficient intensity. Before taking a sample, the sampling pipe must be purged of the gases already contained in it for a period of at least 5 minutes and then connected to the sample container by a short length of rubber tubing. The purging of the sampling pipeline may, however create a dangerous atmosphere for the sampler. If the sampler is not equipped with a respiratory protective apparatus, he should carry a carbon monoxide detector and an oxygen-measuring instrument or a flame safety lamp with him.

When a sampling pipe is drawing in or when there is no pressure in it, a footpump, a Fletchers bellows, or a compressed-air ejector, where a compressed-air line is available, may be used to purge the sampling pipe of gases and to draw the sample into a glass or plastic sample container (Fig. 5.5). For collecting sample in a brass capsule, the suction nipple of the pump is connected by a short piece of tubing to the cock as in the arrangement shown in Fig. 5.6.

Sampling the atmosphere behind a stopping may have to be carried on for a period of a few months or longer.



length of rubber tubing which should be held in still air. A differential automatic pressure recorder may also be used.

Barometric pressure

The barometric pressure should be observed and recorded whenever samples from a sealed-off area are collected. Knowledge of the barometric pressure may help in explaining the make of gases, particularly oxygen and firedamp, in the sealed-off area.

Air quantity

Measurement of air quantity flowing past a sampling place assists in interpretation of the results of sampling. Where there is measurable air movement, an anemometer or other air-velocity measuring instrument may be used. In nearly-stagnant or slowly-moving air, a chemically-produced smoke may be used to estimate air-flow rate.

Tables 5.1 and 5.2 show specimen cards or pages of a Mine Atmosphere Sample Record and Mine Atmosphere Sample Analysis Record, respectively, cord

teorological data				
ound	Surface		Sampler	
rom. essure rr/mbar	Temp. °C	Pressure Torr/mbar	Name & designation	Remarks

record

Results of analysis											Remarks
CO ppm	H_2 %	C_nH_n	CO_2 %	O_2 %	N_2	% total	CO index	Initials	Initials	Initials	

5.3.1 Status of fire

The following gas concentration ratios that are unaffected by dilution have found widespread acceptance over the years as criteria of the state of heatings and fires in their early as well as in the later stages of development:

$$\text{CO index } \frac{CO}{\Delta O_2} = \frac{100\,CO}{0.265\,N_2 - O_2}$$

Jones–Trickett (JT) ratio [77]

$$\text{JT ratio} = \frac{CO_2 + 0.75\,CO - 0.25\,H_2}{0.265\,N_2 - O_2}$$

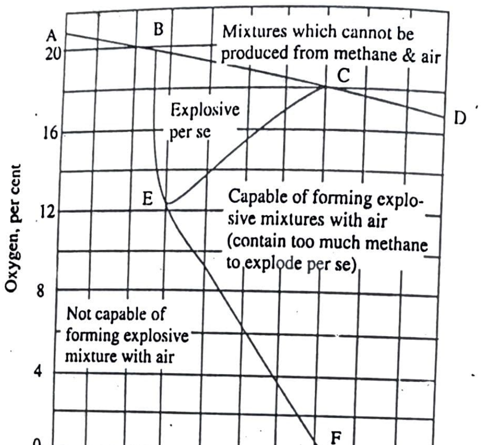
It should be pointed out that the CO index is not to be completely relied upon as

limits, upon the development, extent, and intensity of fire at the time of sealing as well as upon dilution through methane and leakage air, consists in general of CO, O_2, CO_2, N_2, CH_4, seldom H_2, H_2S, and C_nH_{2n}. Table 5.3 gives the range of composition of mine firegases [78].

A given mixture of firegases can be considered as consisting of combustibles, suppressors of combustion, and oxygen. It is the combustible constituents of the firegases, notably methane, that combined represent an explosion hazard in a fire area during its sealing or unsealing. The presence of small quantities of the combustible gases CO, H_2 and C_nH_{2n} has the effect of reducing the lower limit of flammability of methane–air mixtures. Table 5.4 gives the flammable limits of the combustible gases mixed with air [78].

Table 5.3 Composition of mine firegases

Constituent	From about Vol. -%	to about Vol. -%
Methane (CH_4)	0.2	80
Carbon monoxide (CO)	0.001	2



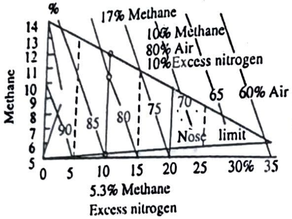
Coward triangles for gases other than methane can also be drawn. Figure 5.8 shows the diagram with flammability triangles for CH_4, CO, and H_2.

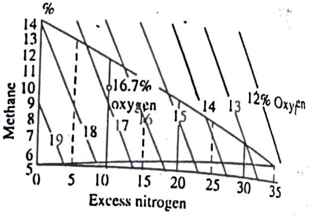
5.3.2.2 Drekopf Method

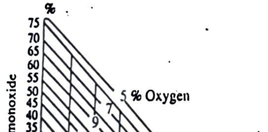
Drekopf [78] reduces the composition of mine firegases as found from chemical analysis to form combustible constituents, air, and excess nitrogen and uses the explosion-suppressing excess nitrogen as a measure of determining the flammability of a given mixture of firegas.

Drekopf represents graphically:

(1) the flammability of mixtures consisting of one combustible gas, air, and excess nitrogen;
(2) the flammability of mixtures consisting of two combustible gases and air;
(3) the flammability of mixtures consisting of two combustible gases, air and excess nitrogen;
(4) the flammability indices for mixtures consisting of two combustible gases, air, and excess nitrogen; and
indices for mixtures consisting of four combustible gases,







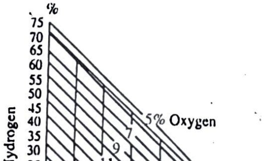


Table 5.5 gives the percentages of the combustible gases and the excess nitrogen at the nose limits and the inert factors.

Table 5.5 Nose limits of gases, excess nitrogen and inert factor

Combustible gas	Nose limit of gas. Vol.%	Excess nitrogen Vol.–%	Inert factor
Methane (CH_4)	5.93	36.6	6.00
Carbon.monoxide (CO)	13.78	57.2	4.15
Hydrogen (H_2)	4.30	71.2	16.55
Unsat. hydrocarbons (C_nH_{2n})	3.10	47.7	15.40

Flammability of mixture of two combustible gases and air (Fig.5.10)

The different diagrams for mixtures of firegases in which two or more combustible

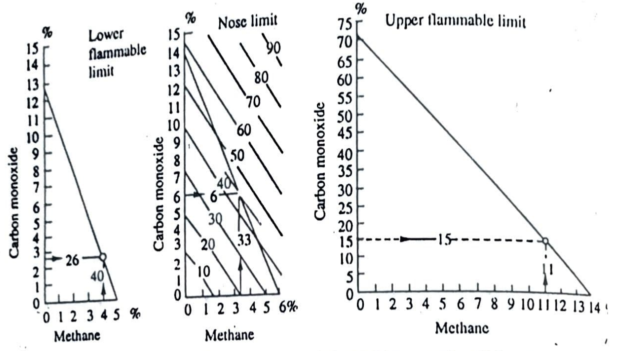


Figure 5.10 Flammability of mixtures consisting of two combustible gases and air.

When $K < 1$, the mixture of firegases lies below the lower limit of flammability

---

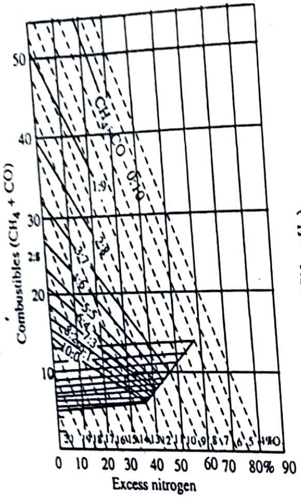
$$\frac{p_1}{14} + \frac{p_2}{72} = 1 \text{ (at the upper limit of flammability)}$$

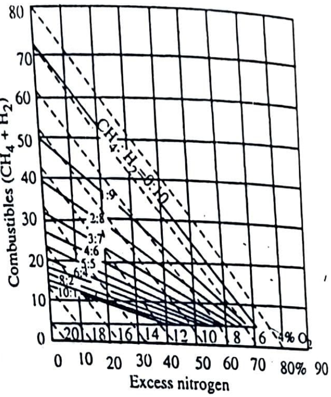
where p_1 and p_2 are the percentages of methane and carbon monoxide in the mixture respectively.

Flammability of mixtures of two combustible gases, air, and excess nitrogen. The graphs (Fig. 5.11) are drawn by using the relations:

$$l_m = \frac{p_1 + p_2}{\frac{p_1}{l_1} + \frac{p_2}{l_2}} = \frac{\frac{p_1}{p_2} + 1}{\frac{1}{l_1} \cdot \frac{p_1}{p_1} + \frac{1}{l_2}}$$

$$s_m = \frac{p_1 + p_2}{\frac{p_1}{} + \frac{p_2}{}} = \frac{\frac{p_1}{p_2} + 1}{\frac{1}{} \cdot \frac{p_1}{} + \frac{1}{}}$$





$$A_s + B_s = K_s \text{ where } A_s = \frac{p_1}{s_1} + \frac{p_2}{s_2},$$

$$\text{and} \qquad B_s = \frac{p_3}{s_3} + \frac{p_4}{s_4},$$

$$A_n + B_n = K_n \text{ where } A_n = p_1 f_1 + p_2 f_2,$$

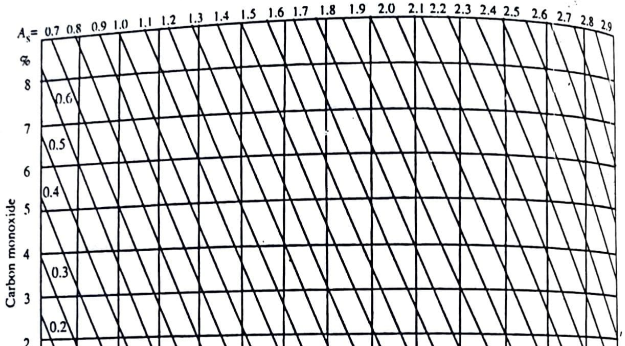
$$\text{and} \qquad B_n = p_3 f_3 + p_4 f_4,$$

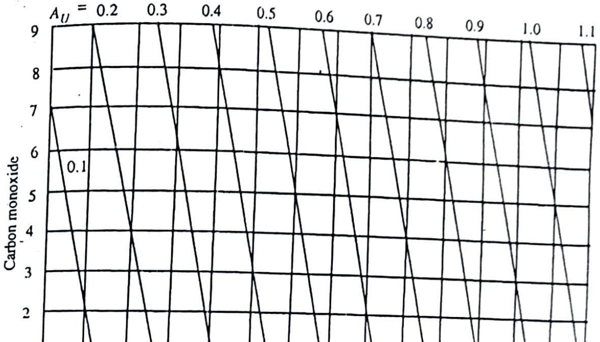
$$A_u + B_u = K_u \text{ where } A_u = \frac{p_1}{u_1} + \frac{p_2}{u_2},$$

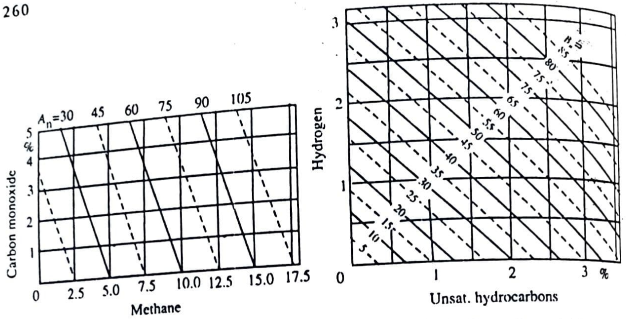
$$\text{and} \qquad B_u = \frac{p_3}{u_3} + \frac{p_4}{u_4},$$

The A- and B- values for a given result of analysis can be obtained from the diagrams (Figs. 5.12, 5.13, 5.14 and 5.15) prepared by using the above relations.

When using the diagrams, one finds the values of A_s and B_s from the knowledge of the percentages of the combustible gases and the obtains $K_s = A_s + B_s$


A_S= 0.7 0.8 0.9 1.0 1.1 1.2 1.3 1.4 1.5 1.6 1.7 1.8 1.9 2.0 2.1 2.2 2.3 2.4 2.5 2.6 2.7 2.8 2.9
%
8
7
6
5
4
3
2
Carbon monoxide
0.6
0.5
0.4
0.3
0.2


A_U = 0.2 0.3 0.4 0.5 0.6 0.7 0.8 0.9 1.0 1.1
9
8
7
6
5
4
3
2
Carbon monoxide
0.1



Diagrams for determining the flammability indices A_u and B_u.

The dilution factor, k_u, indicates how far the original mixture of firegases must be diluted in order to bring it to the upper limit of flammability. It is determined by using the relation

$$k_u = K_u + \frac{N_e}{K_n}(K_s - K_u)$$

under the condition $k_u \leq K_s$.

In the above two equations, the dilution factors k_l and k_u as well as the excess nitrogen are variable quantities. These equations represent in rectangular coordinates with the ordinates (K_u, K_s, K_l, K_u, and k_l) and abscissae (N_e, K_n) two straight lines which form a triangle ABC called the Characteristics Triangle (Fig. 5.16). The sides AB and CB correspond to the upper and lower limits of flammability of the diluted firegases, respectively.

When using the diagram, the following points must be kept in mind:

5.3.2.3 Hughes and Raybould method

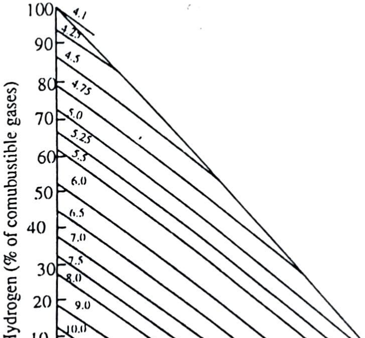
This method [80] utilizes limit diagrams in order to construct an explosive triangle for a given mixture of mine firegases containing methane, hydrogen, and carbon monoxide. The very small amount of unsaturated hydrocarbons is neglected, and carbon dioxide is equated with nitrogen. The flammability characteristics of the given mixture are then determined by plotting its composition on the explosive triangle.

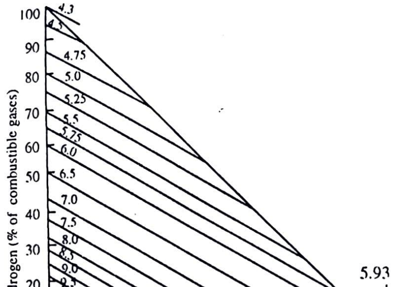
The limit diagrams for the combustible gases are constructed using the Le Chatelier formula. They represent:

(1) The lower limit of flammability (combustible content) of mixtures, consisting of three combustible gases and air (Fig. 5.17);

(2) The oxygen content at the nose limit of mixtures consisting of three combustible gases, air, and excess nitrogen (Fig. 5.18);

(3) The combustible content at the nose limit of mixtures consisting of three





$$\frac{p_1}{s_1} + \frac{p_2}{s_2} + \frac{p_3}{s_3} = \frac{100}{s_m}$$

Construction of the diagram shown in Fig. 5.18

Percentage of excess nitrogen at nose limit

$$n_s = \frac{p_1 f_1 + p_2 f_2 + p_3 f_3}{\dfrac{p_1}{s_1} + \dfrac{p_2}{s_2} + \dfrac{p_3}{s_3}}$$

Substituting the values of the inert factors for CH_4, H_2, and CO as well as their nose limits as given by Drekopf,

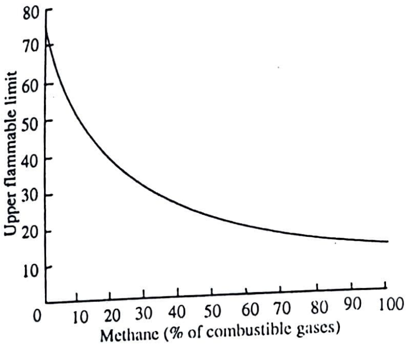


Figure 5.20 Diagram for determining the UFL of mixtures consisting of three

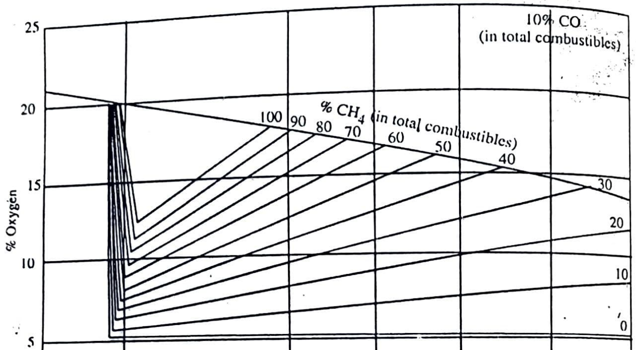
$$M = \frac{\%CH_4}{\%CO + \%H_2 + \%CH_4}$$

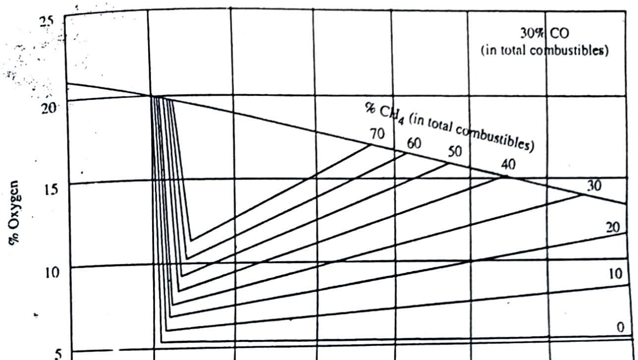
$$L = \frac{\%CO}{\%CO + \%H_2 + \%CH_4}$$

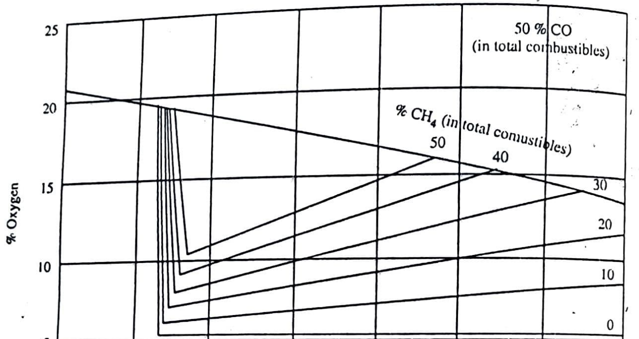
A set of Coward triangles is constructed for mixtures containing, 0, 10, 20, 30, 40, and 50 per cent CO in the total combustible (*L*)for a selected range of values of *M* which will vary from diagram to diagram (Figs. 5.21 to 5.26).

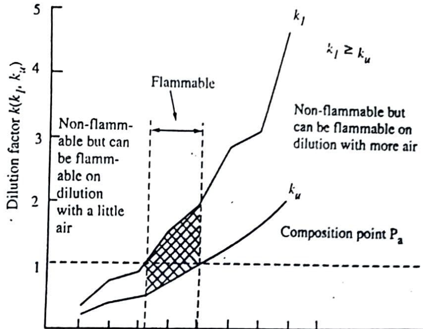
To use the set of diagrams, the ratios *L* and *M*, the percentage of total combustibles, and the percentage of oxygen are calculated from the gas analysis. The diagram with the nearest value of *L* is chosen and the triangles interpolated for the required *M* value. The point corresponding to the composition of the atmosphere (percentage total combustibles, percentage oxygen) is plotted on the diagram and the line joining the 100 per cent air apex (that is 20.93 per cent O_2) to the point is then drawn. The points where the line would intersect the

give the percentages of combustibles and oxygen at the


25
20
15
10
5
% Oxygen
10% CO
(in total combustibles)
% CH_4 (in total combustibles)
100
90
80
70
60
50
40
30
20
10
0


25
20
15
10
5
% Oxygen
30% CO
(in total combustibles)
% CH_4 (in total combustibles)
70
60
50
40
30
20
10
0


25
20
15
10
% Oxygen
50 % CO
(in total combustibles)
% CH$_4$ (in total comustibles)
50
40
30
20
10
0


5
4
3
2
1
Dilution factor $k(k_l, k_u)$
k_l
$k_l \geq k_u$
Flammable
Non-flammable but can be flammable on dilution with a little air
Non-flammable but can be flammable on dilution with more air
k_u
Composition point P_a

CHAPTER 6

Reopening of Sealed-off Areas in Mines

burn faster and are more likely to be rekindled than coals having a lower ratio. The nature of the adjacent strata, especially when they are combustible, influences the time of reopening as they retain heat for a long time even after the oxygen content has fallen below the limit when combustion ceases. Valuable information can be obtained by the use of thermocouple in the sealed area. Sub-micrometre particle concentrations in the sealed area have been found to correlate exceptionally well with the temperatures within the area, independent of the combustibles involved in sharp contrast to the gas concentration data which exhibit no correlation with the temperatures, and their measurement may be used as a more universal technique for determining the status of the underground sealed fire area [83].

3. *Airtightness of stoppings and the sealed-off area*

Airtightness of stoppings and the enclosed as it is nearly impossible to predict accurately the air leakage, the rate of methane emission, and the reliability of the gas samples as representative of conditions in the vicinity of the fireseat.

5. *Original cause of fire*

If the fire was due to spontaneous combustion, rekindling of the fire may take place and contingency plans should be drawn up to meet the situation. A fire caused by machinery would be considered in a different light.

6. *Inert gas injection*

The time of reopening a sealed area can be considerably reduced and the area may be safely entered by injection of an inert gas into it.

6.2 Prepartory Measures for Reopening

3. Time allowed for cooling off;
4. Volume of firedamp enclosed;
5. Probable condition of doors and other ventilation appliances in the area;
6. Inclination of the roadways;
7. Accessibility of roadways;
8. Presence of water in the area;
9. Temperature; and
10. Availability of an inert gas for atmospheric inerting.

6.3.1 Recovery by reventilation

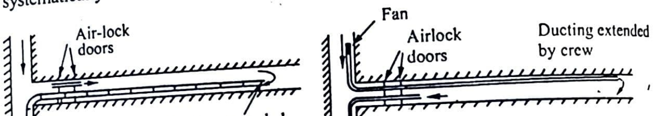
There are two variants of this method:

(a) Reventilation without inspection by rescue brigades (direct method);
(b) Reventilation following inspection by rescue brigades.

spread out and transported away. If during reopening smoke begins to be formed, the fireseat should be closely sealed off by stoppings.

Where an adequate supply of inert nitrogen gas is available, it may be used, instead of air, to flush out the gases.

When in an area being reopened, there are heavy falls in roadways which cannot be easily cleared and re-ventilation in the normal way is not practicable, recourse must be taken to auxiliary ventilation using a brattice line or ducting with or without a mechanical ventilator. A fresh-air base is established and the rescue brigade advances systematically away from it by extending the brattice line or ducting.



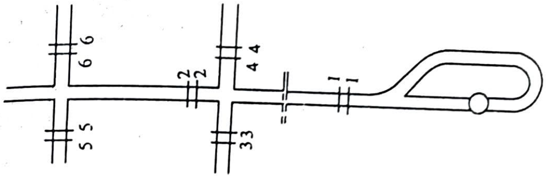


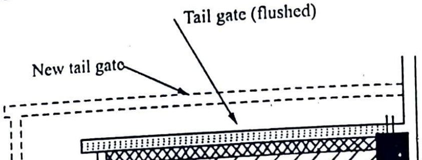
Figure 6.2 Stage method by means of air-locks.

compressed-air lines, if existing, are extended into it. After the air-lock has dried, a rescue brigade, with another fully equipped in reserve at the fresh-air base, enters the area through the air-lock, breaches the stopping, and inspects up to a point where the next air-lock is to be erected, a distance generally lying between 50 and 150 m, depending on the difficulties encountered. The exploring brigade should observe the general conditions, collect air samples, take hygrometer readings,

6.3.3 Other methods

Reopening of hydraulically-flushed sealed district

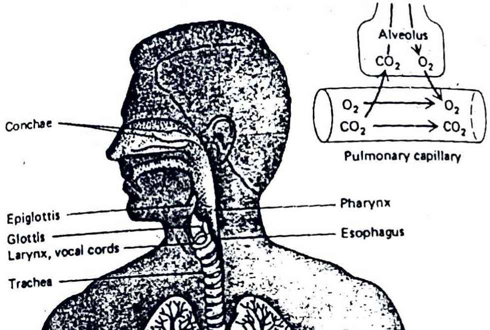
When a fire is sealed off with stoppings in the usual manner and the fireseat or the fire area is flushed with sand or clay, the district can be reopened either by cleaning the mine workings of the flushed material or driving new ones to start a new district (Fig. 6.3). This method is suitable for smaller fires.



CHAPTER 7

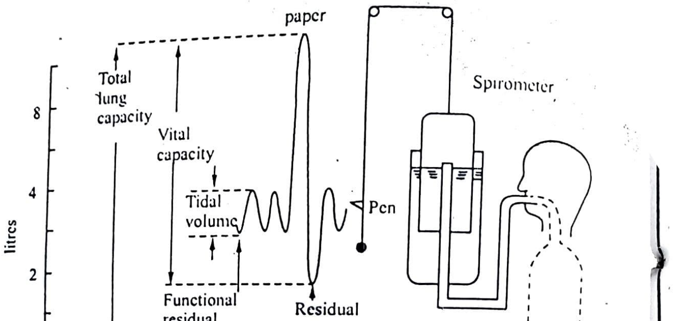
Mine Rescue

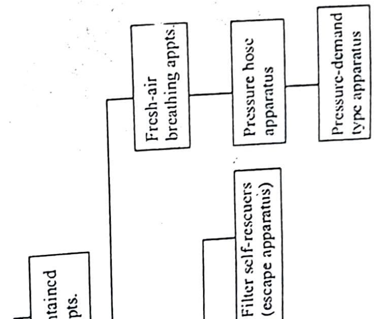
The subject of mine rescue deals with all measures and devices used for working in irrespirable and toxic atmospheres occurring in mines for the purpose of fire-fighting and sealing off mine fires and rescue and/or recovery operations speedily and safely. The saving of lives of men overcome by toxic gases or oxygen-deficient atmospheres forms the immediate and the most important objective of mine rescue.


Alveolus
CO$_2$
O$_2$
O$_2$
CO$_2$
O$_2$
CO$_2$
Pulmonary capillary
Conchae
Epiglottis
Glottis
Larynx, vocal cords
Trachea
Pharynx
Esophagus

inspiratory reserve volume. The amount of air remaining in the lungs at the end of normal expiration (about 2300 ml) is the *functional residual capacity,* (FRC); it equals the expiratory reserve volume plus the residual volume. The maximum amount of air that a person can expel from the lungs following the deepest inspiration is called the *vital capacity,* VC (about 4600 ml), and is made up of the ispiratory reserve volume, the tidal volume, and the expiratory reserve volume; it varies in different persons depending on the age, health, and respiratory training and is about 3 to 4 litres for rescue-trained men. The *vital capacity* is related to the surface area of the body and amounts to 2.6 litres per m^2 surface area. The *total lung capacity* (TLC), is the maximum volume to which the lungs can be expanded with the deepest inspiration (about 5800 ml); it is equal to the sum of the vital capacity and the residual volume.

The interrelationships between the different pulmonary capacities and volumes can be described as follows:


paper
Spirometer
Total lung capacity
Vital capacity
Tidal volume
Pen
Functional residual
Residual
litres
8
4
2


Fresh-air breathing appts.
Pressure hose apparatus
Pressure-demand type apparatus
Filter self-rescuers (escape apparatus)
tained
pts.

The European Committee for Standardization comprising the national standards institutes of eighteen States (Belgium, Denmark, Germany, Finland, France, Greece, Ireland, Iceland, Italy, Luxemberg, Netherland, Norway, Austria, Portugal, Sweden, Spain, and the United Kingdom) laid down stringent European Standards or Norms for compliance in the design and testing of all types of respiratory protective equipment for use in toxic, oxygen-deficient or immediately-dangerous-to-life-or-health atmospheres in mines. The European Standards stipulate the minimum requirements to be fulfilled in the design, materials of construction, and performance testing of not only the whole apparatus but also their component parts. The tests include tests conducted using a breathing machine and man-tests conducted by specially trained personnel under different work load conditions usually encountered in practice. In the USA, the Mine Safety and Health Administration and the National Institute for Occupational Safety and Health jointly approve the design and certification of respirators as a part of product approval.

Depending on the source of oxygen supply, the apparatus may be divided into:

(1) Compressed-oxygen apparatus;
(2) Self-generating or chemical oxygen apparatus; and
(3) Liquid-oxygen apparatus.

Compressed-oxygen apparatus are the principal equipments used by all mine rescue stations throughout the world.

In the open-circuit apparatus, oxygen or air from a compressed-oxygen or compressed-air reserve, is supplied to the wearer and the exhaled oxygen or air is discharged into the open and is put to no further use. They are also called reservoir or non-regenerative apparatus.

All designs of the closed-circuit- type apparatus are essentially the same as those of the open-circuit type. They differ only in construction of the several parts, source of oxygen or air, and the arrangement of the parts and the circulating

(5) *Compactness*

The apparatus should be designed to be as compact as possible and unnecessary projections are eliminated so that work in intricate and confined spaces is facilitated. The advantages of a modular component design which facilitates servicing and maintenance should not be lost sight of.

(6) *Durability*

The apparatus should be of robust construction. Before being officially approved for use in mines, all individual parts of the apparatus as well as the assembled apparatus must pass laboratory and practical approval tests which vary in rigorousness from country to country.

(7) *Initial cost and ease and cost of maintenance.*

they have to be brought through irrespirable and poisonous atmospheres. They have an oxygen reserve of 120, 150 or 200 litres.

Depending on the type of oxygen feed or dosage, compressed-oxygen apparatus can be classified into the following three groups:

(a) Apparatus with a constant dosage supplemented, when necessary, by manual operation of a bypass valve.
(b) Apparatus in which the dosage is entirely lung-governed.
(c) Apparatus with a constant but small dosage of oxygen supplemented when necessary by a lung-governed oxygen admission valve (partly or semi-lung-governed apparatus).

In apparatus with a constant dosage, oxygen at a definite predetermined rate, 2 litres per minute, is supplied to the wearer when he is working hard but when

Siebe-Gorman Mark IV and Mark V Proto apparatus in which the absorbent is placed at the bottom of the breathing bag. As a regenerating chemical, alkali (anhydrous sodium hydroxide), lithium hydroxide, and soda lime (a mixture of sodium hydroxide and calcium hydroxide) are used. Extensive comparative tests with regenerators filled with alkali and soda-lime conducted by the Draegerwerk, Lübeck [84] showed that, although the efficiency of absorption of carbon dioxide can be considered the same for the two chemicals, alkali has a decisive advantage over soda-lime in that the dry and wet bulb temperatures of the inspired air with apparatus using alkali are significantly lower than with apparatus using soda lime. Investigations by the US Bureau of Mines [85] showed that LiOH has the greatest capacity for CO_2 absorption as compared to NaOH or soda lime.

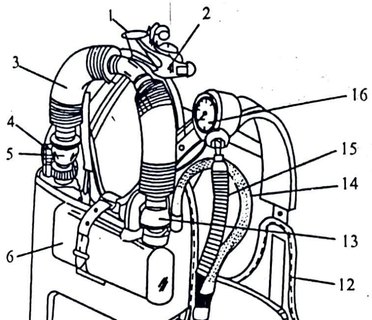
A breathing bag to serve as an air reservoir from and into which the wearer breathes. The bag also provides flexibility to the circulatory system. It is made of high-grade sheet rubber or rubberized fabric, gas-impervious, and usually has one compartment with a useful capacity of 4 to 6 litres. Exception, however, is the

Inhalation (inspiratory) and exhalation (expiratory) valves to compel the circulating air to travel in one direction. The valves are usually of spring-loaded mica-disc type or of neoprene rubber type and are extremely sensitive. They are located in the circulatory system differently by different manufacturers. In the Draeger apparatus, they are located inside the apparatus casing.

A pressure gauge to indicate to the wearer the available supply and duration of oxygen in the oxygen cylinder. It is built into a strong brass case and is mounted on a special flexible high-presure tube attached to the outlet of a pressure-gauge valve which is provided on the high-pressure fitting of the reducing valve.

A bypass valve to supply oxygen direct to the wearer should the reducing valve or the lung-governed admission valve through any cause fail to act properly. The valve is usually designed as a self-closing push-button valve.

A lung-governed oxygen admission valve or regulator in the case of automatic lung-governed apparatus to supply the wearer with oxygen in the exact quantity which his exertion may demand. There are two types of lung-governed



to a little over the bottom and a charge of 2 kg of carbon dioxide absorbent 'Protosorb' is placed at the bottom of the bag divided equally between the front and rear compartments. It is fitted at the top right-hand corner with a manual relief valve.

(2) A cooler is used to reduce the temperature of the regenerated air before inhalation. The cooler is a two-compartment copper chamber containing 170 g of a chemical disodium phosphate which melts at a temperature of 35°C. The melting of the sodium phosphate results in absorption of heat. The chemical resolidifies on cooling and remains permanently in the cooler.

(3) The reducing valve is of the well-proven diaphragm design in which a spring-loaded diaphragm is balanced by the reduced pressure at which the reducer is set so that the diaphragm is continually moving backwards and

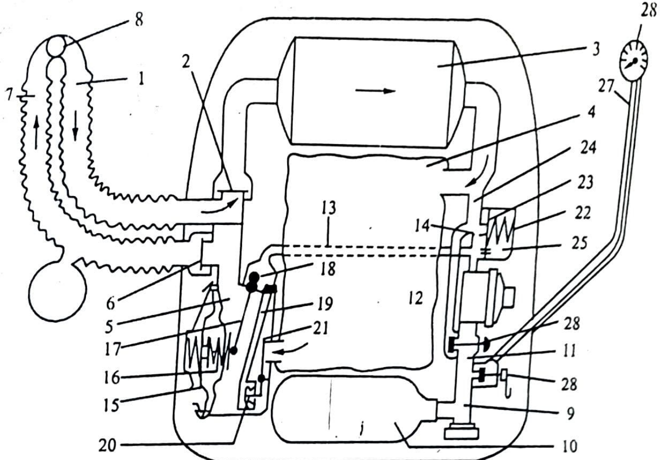
The Proto breathing apparatus which had been in use in the Indian mining industry for mine rescue work over six decades has become obsolete and is being increasingly replaced by pressure-demand type in which the pressure inside the system in relation to the environment is positive during both inhalation and exhalation.

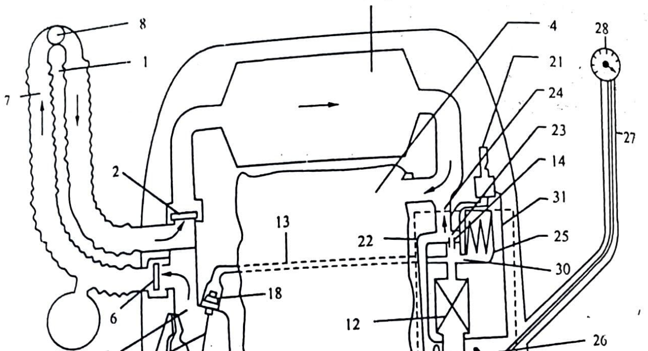
7.2.1.2.3 Draeger Model BG 174 Apparatus (Fig. 7.7)

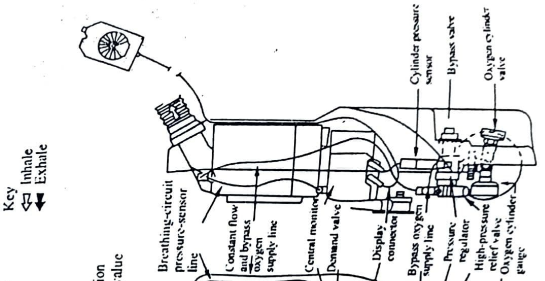
This apparatus manufactured by Draegerwerk AG, Lübeck, Germany, is widely used as a standard mine rescue apparatus in many countries of Europe and India. It is of the closed-circuit type with a constant and lung-governed dosages or feeds of oxygen. It was introduced in India in 1992 to replace the Proto Mark IV apparatus.

Specifications

Endurance (nominal)	4 hours hard exertion at a working pressure of 200 bar
Cylinder capacity, empty	2 litres
Cylinder capacity at 200 bar	400 litres
Breathing bag capacity, full	6 litres




3
8
1
7
2
4
21
24
23
14
31
25
30
22
13
18
6
12
26
27
28


Key
Inhale
Exhale
Cylinder pressure sensor
Bypass valve
Oxygen cylinder valve
Breathing-circuit pressure-sensor line
Constant flow and bypass oxygen supply line
Central monitor
Demand valve
Display connector
Bypass oxygen supply line
Pressure regulator
High-pressure relief valve
Oxygen cylinder gauge
ion
alue

ssed-oxygen breathing *apparatus*

7.2.1.2.4 Drager BG 4 Apparatus (Fig. 7.9)

Draeger BG 4 self-contained compressed-oxygen breathing apparatus (Fig.7.10) is a technically-advanced model of BG 174, which uses a disposable soda lime canister and a water-ice cooling unit to cool the inhaled air. It is approved by NIOSH CFR 30 and EN- 145 European Standard. It was introduced in India in 2002.

Specification:

Endurance	4 hours hard exertion at a working pressure of 200 bar
Cylinder volume, empty	2 litres
Breathing bag volume	5.5 litres
Oxygen flow rate	
constant dosage	1.5 to 1.8 litres per minute
lung-governed dosage	minimum 80 litres per minute

Reliable information on the status of the apparatus during use is provided by the Monitron which is an electronic pressure monitor with audible and visual alarms. The Monitron is automatically activated as soon as the cylinder valve is opened. It automatically performs a self-test and a battery check. By opening and closing the cylinder valve, the user can perform a high-pressure leak test in less than 15 seconds. When the pressure falls during operation, it flashes a red warning light. It sounds a residual pressure alarm for 30 seconds when the cylinder oxygen pressure is about 48 bar (700 psi) and again continuously when the pressure is 10 bar (145 psi).

Circulatory System

The exhaled air containing carbon dioxide and excess nitrogen flows through the exhalation valve, exhalation tube into the soda-lime absorption canister or scrubber where it is freed from carbon dioxide generating heat in the process. The air, thus purified, flows into the breathing bag and, on inhalation, flows

The salient features of the apparatus are:

(1) High- and low-oxygen flow-rates or dosages which can be set at 5 litres per minute for 2-hour duration and at 10 litres per minute for one-hour duration, depending on the environmental temperature. The high flow-rates together with the incorporated evaporative cooling system reduce the inspired air temperature under hard working conditions from the normal 45°–50°C to 39°C at 5 litres per minute flow-rate and to 35°C at 10 litres per minute flow-rate. It has no demand valve.

(2) Elimination of pre-start-up purging of the apparatus by incorporation of a spring-loaded breathing bag diaphragm contained in a gas-tight chamber. When the apparatus is not in use, the breathing bag is held deflated by the spring and, consequently, it is possible to introduce oxygen immediately into the breathing bag and thus to the wearer without any need for purging of the breathing circuit.

(3) The breathing bag diaphragm can be flushed with fresh oxygen by means of a bypass valve fitted on the pressure reducing valve.

(4) A heat exchanger unit containing copper wire windings, which is covered with a wet cloth. The heat exchanger functions in tandem with the ambient-air chamber containing the breathing bag. Ambient air is drawn over the

(5) The CO_2-absorbent canister contains 2 kg of soda lime, size 6 to 10 mesh, and is of split-bed design.

(6) Low breathing resistance compared with other self-contained oxygen breathing apparatus.

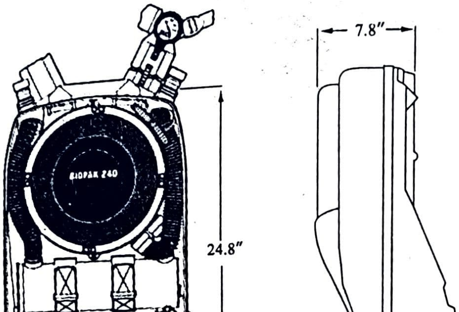
(7) Simplicity of construction and ease of maintenance.

(8) For high temperature work, the flow of oxygen can be doubled halving the working duration to one hour but cooling the set further.

(9) The apparatus is housed in a stainless steel casing.

Circulatory system

The exhaled air containing unused oxygen, CO_2, and moisture flows through the exhalation valve, corrugated exhalation tube into the purifying canister. The air thus purified but hot then passes through the heat exchanger, breathing bag diaphragm and inhalation valve into the inhalation tube where it gets enriched with fresh oxygen before it is drawn into the wearer's lungs. During inhalation, the breathing bag diaphragm collapses, pressure drops in the surrounding ambient-air chamber and the non-return valve opens to fill the chamber with the ambient air. On exhalation, the breathing bag volume increases the pressure inside the chamber rises, the non-return valve closes, the non-return valve in the wetted

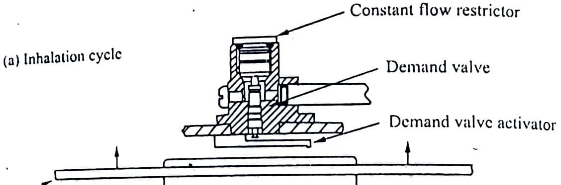


3. Carbon dioxide absorption canister with LimePak CO_2-absorbent which considerably reduces the pre-use cost compared with other SCBAs.
4. The facepiece of anti-oxidant silicone rubber has an anti-fog insert, a speaking diaphragm, and a CO_2 build-up prevention nose cup.
5. A high-pitched pressure whistle alarm which sounds for 30 to 60 seconds when 25% of the rated duration remains in the cylinder (approx. 60 min).
6. The weight of the apparatus is carried on the hip. Over-the-shoulder design of neoprene hoses with low breathing resistance reduces wearer fatigue.
7. The oxygen cylinder is fibre glass-wrapped aluminium cylinder.

Specifications:

Endurance	4 h
Cylinder capacity (empty)	2.76 litre
Cylinder capacity at	

(1) It provides cool dry oxygen to the wearer at a temperature below the ambient temperature for full duration of the apparatus;
(2) It removes heat from the wearer's body via respiratory tract;
(3) Breathing resistance with high flow-rate is low so that it is hardly noticeable even under heavy work conditions;

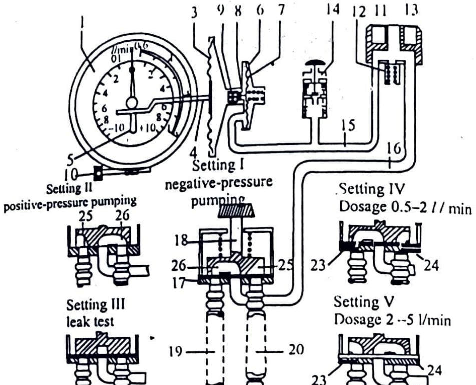


(4) It is free of high-pressure components and requires less maintenance;
(5) It is automatic in use;
(6) It is simple to maintain.

Aerorlox, the liquid oxygen breathing apparatus manufactured by SiebeGorman, UK, was inducted in the British mining industry in the late 1960s as standard breathing apparatus for mine rescue stations but it was superseded by SEFA apparatus in 1989 due probably to the dangers and difficulties associated with the handling of the liquid oxygen and lack of flexibility in its use.

7 2.1.4 Self-contained non-regenerative or reservoir apparatus

In this open-circuit-type of apparatus, the inhaled air is supplied from a pressure gas cylinder containing air or oxygen at 200 or 300 bar pressure and the exhaled air passes through an exhalation valve into the outside atmosphere. Such apparatus


1
3
9
8
6
7
14
12
11
13
l/min 0.6
01
2
4
6
8
-10
+10
15
16
4
5
10
Setting I
negative-pressure
pumping
Setting II
positive-pressure pumping
25
26
18
26
17
25
Setting IV
Dosage 0.5–2 l / min
23
24
Setting III
leak test
19
20
Setting V
Dosage 2 --5 l/min
23
24

(9) Breathing bag volume test;
(10) Bypass test;
(11) Whistle activation and duration tests; and
(12) High-pressure leak test.

The test programme for self-contained compressed-air breathing apparatus comprises:

(1) Leak tests at positive and negative pressures;
(2) Lung demand valve test for response pressure; and
(3) Exhalation valve test.

The test programme of self-contained chemical oxygen apparatus includes the following:

(1) O_2-content in the inspired air;
(2) Temperature of the inspired air;
(3) Inhalation resistance;

The levels of the physiological parameters considered to be acceptable were:

Average inhaled CO_2	<	4%
Average inhaled O_2	>	15%
Wet-bulb temperature	<	50°C
Exhalation pressure	<	200 mm WG
Inhalation pressure	>	300 mm WG

Comparison of performance test results showed that:

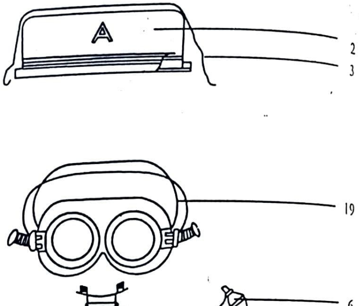
1. The Draeger BG 174A and the BG 4 have the highest durations.
2. The BG 4 has the highest level of positive pressure and the lowest pressure swing (exhalation and inhalation pressures combined) with Sabre SEFA coming a close second.

$$2K_2O_4 + 2CO_2 \longrightarrow 2K_2CO_3 + 3O_2$$

$$2K_2O_4 + 2H_2O \longrightarrow 4KOH + 3O_2$$

$$2KOH + CO_2 \longrightarrow K_2CO_3 + H_2O$$

The rate of liberation of oxygen is governed by the rate of breathing by the wearer under various exertion levels, the degree of training, and the condition of the apparatus. Once the chemical reaction has started, oxygen is liberated faster than it is consumed until the depletion of KO_2 and, in consequence, a pressure is built up in the circulatory system of the apparatus unless the excess oxygen is released by means of a relief valve. The apparatus, when worn, has, therefore, a slight positive pressure in all parts of its circulatory system. All chemical



breathing tube, and a valve housing incorporating a particulate filter, inhalation and exhalation valves, a heat exchanger, and a relief valve. The heat exchanger maintains the temperature of the inhalation air below 50°C. The oxygen produced by the chemical is inhaled through the breathing bag, filter and heat exchanger.

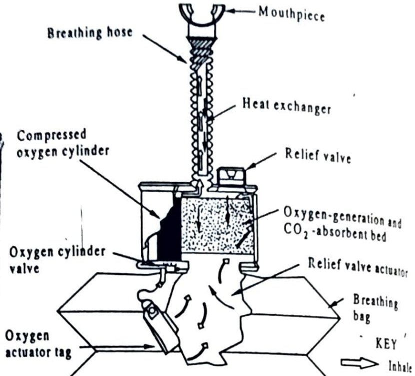
The technical specifications of the SSR 90 are:
Dimensions (with container):

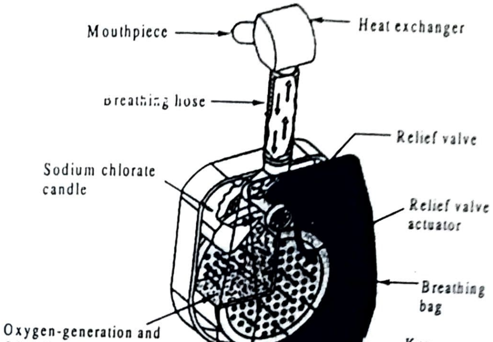
Height	310 mm
Width	220 mm
Depth	120 mm

Weight:

With container	4.9 kgf

Moisture indicator
Moisture indicator


Mouthpiece
Breathing hose
Heat exchanger
Compressed oxygen cylinder
Relief valve
Oxygen-generation and CO_2-absorbent bed
Oxygen cylinder valve
Relief valve actuator
Breathing bag
KEY
Oxygen actuator tag
Inhale


Mouthpiece
Heat exchanger
Breathing hose
Relief valve
Sodium chlorate candle
Relief valve actuator
Breathing bag
Oxygen-generation and

put on or remove the unit without taking off the belt. The carrying case is provided with a colour display which provides a continuous indication of the readiness of the apparatus for use.

The apparatus has NIOSH/MSHA approval for use in the American mines as an escape apparatus.

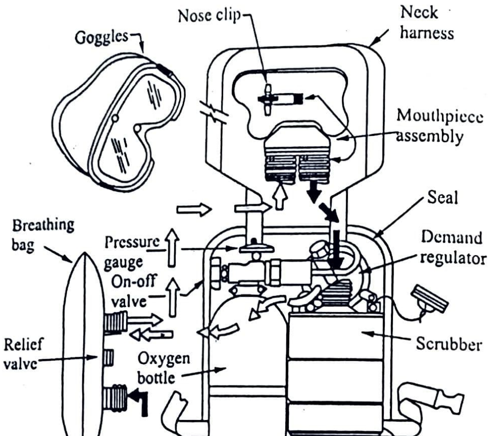
Specifications:

Height	257 mm
Width	200 mm
Depth	105 mm
Weight in case	approx. 2.8 kgf
Weight in use	approx.2.4 kgf
Service life	
at minute volume of	
30 litres per minute	approx. 60 min.
10 litres per minute	approx. 180 min.

under moderate to heavy workload conditions. It can be belt-worn or stored in a cache for emergency evacuation use in toxic or oxygen-deficient atmospheres.

As with other chemical-oxygen self-rescuers, the initial supply of oxygen (approx. 9 to 10 litres) is generated by a chlorate candle which is automatically activated when the apparatus is donned taking about 45 to 90 seconds to fill the circulating system including the breathing bag when it is extended. The self-rescuer consists of a stainless steel housing enclosed in high impact- and heat-resistant urethane. It contains the primary KO_2 bed and the chlorate candle which is the first and immediate source of oxygen. It has a removable end cap on the top of which is a moisture indicator which indicates readiness for use. In the normal air-tight mode, the indicator shows both pink and blue, but, when contaminated, a solid pink thereby eliminating confusion in times of emergency.

The breathing bag is abrasion-resistant with noseclip, mouthpiece, and relief valve all located on it.


Goggles
Nose clip
Neck harness
Mouthpiece assembly
Seal
Breathing bag
Pressure gauge
On-off valve
Demand regulator
Relief valve
Oxygen bottle
Scrubber

Specifications:

Dimensions		
Height	302 mm	
Width	218 mm	
Depth	117 mm	
Weight in use	2.9 kgf	
Cylinder capacity at 200 bar	157 litres	
Breathing bag volume	5 litres	
Oxygen flow rate:		
constant dosage		1.6 litres per minute
demand flow		minimum 30 litres per minute
Operating temperature		minimum – 12.2°C
Storage temperature range		– 12.2°C + 60°C

Service life:

60 min or more during moderate to heavy use

up to 8 hours under rest

Storage life:

Inspection, maintenance and testing procedures are prescribed by the manufacturers and approval sanctioning authorities and every mine should provide for proper inspection, testing, maintenance, and repair by trained persons.

7.2.2 Filtered air respirators, air-purifying respirators, or filter masks

These apparatuses are of the open-circuit type and afford respiratory protection for limited periods against carbon monoxide in atmospheres containing not more than 2 per cent carbon monoxide and not less than 16 per cent oxygen needed to sustain life.

There are two main types of air filter masks that are used in mines:

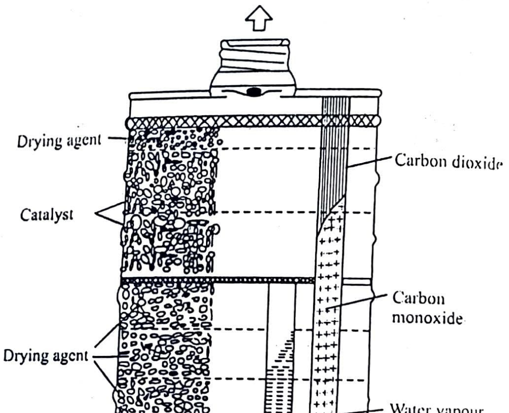
(1) Chemical respirators or filter masks; and
(2) Particulate respirators or mechanical filter respirators.

The chemical respirators can be further divided into:

(1) Canister-type filter masks, gas masks, or gas respirators; and

(i) Cellulose particulate filter for removing dusts, fumes, mists and smoke;
(ii) Impregnated activated charcoal and caustite for absorbing organic vapours and acid gases;
(iii) Activated charcoal and baryta-charcoal for the removal of acid gases;
(iv) Silica gel or charcoal impregnated with sulphuric acid for the removal of ammonia;
(v) Charcoal impregnated with zinc chloride for removal of the bulk of water vapour; and
(vi) Anhydrous calcium chloride or calcium chloride–zinc chloride for removal of final traces of moisture.

The catalytic oxidation of carbon monoxide at ordinary temperatures to the relatively harmless carbon dioxide is effected by 'hopcalite', an activated granular mixture of manganese peroxide and copper oxide which was developed at the John Hopkins University, USA, in 1919. Hopcalite should be specially prepared so that the granules have immediate 'pick-up' (ability to function as soon as the first traces of carbon monoxide come into contact with them), be hard, and show no tendency to powder. The catalytic granules which are hygroscopic



be so great that injury to the canister is likely to occur. Hence, the gas masks should only be used in atmospheres containing not more than 3 per cent volume of carbon monoxide and not less than 16 per cent by volume of oxygen.

The following precautions should be observed when using gas masks in mines:

(1) Only trained men should be allowed to wear the gas mask for entering hazardous atmospheres.

(2) Gas-mask crews should always carry a lighted flame safety lamp or an oxygen meter and a carbon monoxide detector with a measuring range. If it is indicated that the oxygen concentration is less than 16 per cent or carbon monoxide is 2 per cent or more, the crew should immediately return to the fresh-air base.

(3) A man wearing a gas mask should never be allowed to enter a hazardous atmosphere alone without at least one or other man accompanying him.

(4) A crew equipped with a self-contained breathing apparatus should be kept in reserve at the fresh-air base when gas-mask crews advance.

(5) The canister should be changed when its service life has ended or when the

7.2.2.2.1 Acceptance specifications for filter self-rescuers

All filter self-rescuers must be designed to comply with the specifications laid down in respect of duration of protection, performance characteristics, and construction. They are approved for a minimum duration of one hour. The actual duration may be longer or shorter depending upon CO concentration and other environmental conditions. Their safety and performance have steadily improved over the past four decades and the acceptance specifications today are most stringent. The key element in the approval testing is the mechanical-lung machine. The European Standards, EN 404 Class 4 BR, have laid down design and performance guidelines for filter self-rescuers.

The principal German current acceptance specifications are given below:

Breathing resistance (inhalation resistance)

The initial breathing resistance when tested at a constant air flow-rate of 94 litres per minute shall not exceed 65 mm WG (6.5 mbar). When using a test air containing 0.25 per cent CO by volume, the breathing resistance after 60 test minutes shall not exceed 80 mm WG (8 mbar).

Carbon monoxide slip

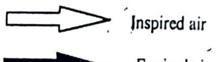
Using the lung machine set at 20 strokes per minute, 1.5 litres per stroke, and a

7.2.2.2.2 Draeger FSR 850

Figure 7.21 shows the sectional view of the canister of the Draeger filter self-rescuer satisfying the German acceptance specifications. It incorporates a heat exchanger in its mouthpiece assembly, which absorbs heat from inhaled air resulting from the catalytic combustion of CO and gives it off to the exhaled air which passes through it and out through the spring-loaded exhalation valve. The drying agent used is activated charcoal impregnated with lithium bromide and silica gel.

The specifications are:

Height	135 mm
Cross-section	94 cm²
Mass	1060 g
Maximum CO-slip using test air containing	

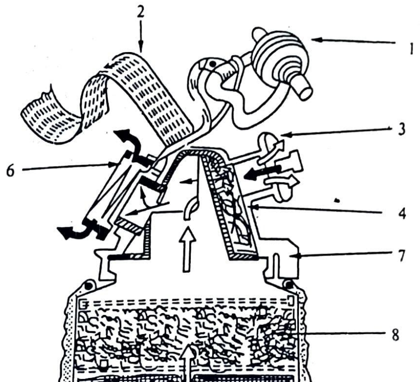


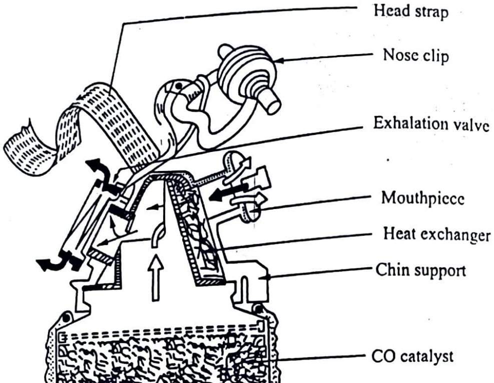
0.25 per cent CO and 20.7 g·water/m³ air after 60 test minutes	0.001%
Inhalation resistance	
unused	6.0 mbar
under test conditions after 60 test-minutes	8.0 mbar
Temperature of inspired air at 1.5 per cent CO, saturated at body temperature	60°C

Joseph Leslie Draeger Manufacturing Ltd., a subsidiary of Draegerwerk, introduced a model IFSR 910 of one-hour duration combining the approved features of Draeger FSR 850 and FSR 950. Draegerwerk has now replaced FSR 950 by Draeger FSR 990.

7.2.2.2.3 MSA/Auer FSR W 65 (Fig. 7.22)

sectional view of the canister of the MSA./Auer filter-rescuer




Head strap
Nose clip
Exhalation valve
Mouthpiece
Heat exchanger
Chin support
CO catalyst

The technical specifications are:

Dimensions (with case)

Height	approx.	140 mm
Width	approx.	100 mm
Depth	approx.	80 mm
Weight with case	approx.	900 g
without case	approx.	520 g

Table 7.2 gives the important characteristics of the new filter self-rescuers Auer W 95 and Draeger FSR 990 compared to the European Standard EN 404 Class 4 BR.

Table 7.2 Important characteristics of the new filter self-rescuers Auer FSR W 95 and Draeger FSR 990 compared with the European Standard EN 404 Class 4 BR [86]

	EN 404 Class 4 BR	Auer FSR W 95	Draeger FSR 990
Dimensions	–	140 mm × 100 mm × 80 mm	151 mm × 138 mm × 90 mm

7.2.2.2.5 Testing and maintenance of filter self-rescuers

All filter rescuers should be inspected and tested periodically to determine their reliability. They should be visually inspected for external defects or damage and air-tightness of seal as well as for air-tightness (where possible) and increase in weight. If a self-rescuer gains in weight more than what is laid down by the manufacturer, it should not be reissued. When determining the weight once a month, it is essential that the self-rescuer is free from dust, dirt, etc., and that no parts of it are missing. The results of tests for air-tightness, nature of any repairs made, and any corrections of weight should be entered in a book kept for the purpose.

The filter self-rescuers are carried on the person and taken off only at the workplace. They are stored on the surface in the lamproom or in a self-rescuer room usually located adjacent to the lamproom. In the self-rescuer room, all units are externally cleaned and visually examined each day to ascertain that they are in safe working condition. Experience has shown that the self-issue system is the best.

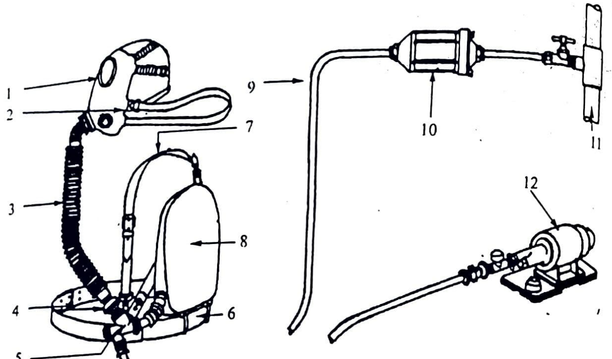
to a swivel-type metal Y-connecting socket on the wearer's back. Such apparatus is particularly suited for work in confined spaces into which the wearer enters from above.

For reasons of safety, suction hose apparatus are not approved for use below ground.

7.2.3.2 Pressure hose apparatus or supplied-air respirators (SAR)

In this type of apparatus, fresh air or breathable air from a clean air source is continuously forced through a hose to the wearer by means of a suitable air-supply device. Such apparatus is used when the distance between the workplace and fresh air is between 20 and 200 m depending on the air-supply means. As the air hose is constantly under a slight positive pressure, there is no danger to the wearer from leakage of outside irrespirable atmosphere into the apparatus around the facepiece.

Pressure hose apparatus is available as lateral- or back-type



The main advantages of an air hose apparatus are its low breathing resistance, longer duration, and the fact that with waist-mounted air cylinders and a mask mounted air-pressure regulator, it can be used, with approval, as a 15-minute stand-alone self-contained breathing apparatus. The disadvantages, however, lie in its heavy large-diameter unwieldy air hoses, restriction of movement, and potential hazard where the hose can come into contact with machinery.

7.2.3.3 Precautions when using fresh-air hose apparatus

The following precautions should be observed when using a fresh-air hose apparatus:

(1) The apparatus should be used by trained rescue men;
(2) Before commencing work, the method of signalling should be decided; if a code of signal is laid down by regulations, it should be followed;
(3) One should never work without an observer who should be a trained rescue man in readiness at the fresh-air source or base for help;

oxygen by artificial respiration and the heart muscles and other important nerve centres must be excited or stimulated to restore the normal blood circulation.

The time factor is of greatest importance for the success of resuscitation. Even while rescuing the patient, the rescuers should constantly think of saving time. There is enough oxygen in the blood to keep cells alive for about 10 minutes but the brain cells start dying after being deprived of oxygen for only 4 minutes. It is not always necessary to take the patient to the fresh-air base but a place in fresh air nearest the place of accident will be satisfactory.

When the patient is to be taken to a hospital on the surface, oxygen should be administered on the way to the hospital. An electrocardiogram and chest X-ray should be taken immediately. Serious cases must be admitted in the intensive care unit where specialised treatment measures can be undertaken.

Resuscitation operation must be continued after it is once started until natural breathing is restored or till such time as the doctors decide that further efforts will be of no avail. There are many accidents in mines which require resuscitation operation. The principal causes of such accidents are:

(a) Throttling of the inspired air due to trapment under the fall of debris,

inhalation but does not remain in the contracted condition during exhalation. The patient must be laid on his back on a flat surface with shoulders raised and supported by a 15-cm thick pad of clothes or folded blanket. All tight clothes around the back, chest, and waist must be loosened.

Ensuring free air passages

In an unconscious person, the mouth is often closed and the tongue falls back and obstructs the air passage. Also, the lower jaw recedes downwards and backwards. To ensure free air passage, therefore, the mouth must be opened by hand or by means of a mouth opener and the pharyngeal cavity cleaned of water, chewing gum, chewing tobacco, mucus, food, blood and also teeth, etc. The tongue must be pulled out either with hand or forceps and held by means of a bandage. In most cases, holding forward of the tongue is not necessary. It is sufficient if the head is tilted backwards and then turned laterally.

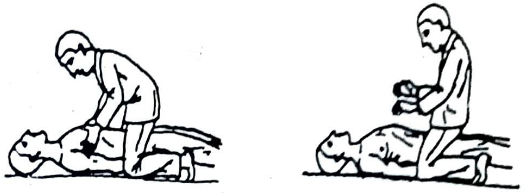
Retaining the body heat

In a completely unconscious person (coma), the formation of heat in the body ceases. For maintenance of metabolism in the body cells, it is timing should be followed. The pressure necessary for respiration should be applied gradually.

Depending on the position in which the patient is laid, manual methods can be further divided into two groups. In one group, the patient is laid on his back (Sylvester and Howard methods) while in the other the patient is laid in the prone position (Holger–Nielsen method).

Sylvester method
It is a two-arm method and is the oldest and the most important method of artificial respiration by hand. It dates back to 1858. It was later modified by Brosch (189 6) and Koch (1924) and is also called Sylvester–Brosch–Koch method [76].

The first-aider or operator kneels just above the patient's head and grasps both his arms immediately below the elbows from outside (Fig. 7.27). He then presses the arms steadily and firmly downwards and inwards against the lower portion of the chest cavity with forearms crossed. The applied pressure is maintained steady for 2 seconds. This causes expiration. Next, the operator grasps the upper arms near the elbows from inside, raises and pulls the arms backwards until they are in line with the body and the elbows are just touching the ground. In this movement, the chest cavity is extended and fresh air is inhaled. A short pause of 2 seconds should be allowed before the expiration


Expiration

Inspiration

Figure 7.28 Howard method of artificial respiration

The Howard method cannot be applied with rib fractures. If the arms are fractured, they should be placed by the side of the body.

Holger–Nielsen back-pressure, arm-lift method
This method dates back to 1932. In this method, the patient lies in the prone position, i.e., face downwards on a flat surface with the head resting on his folded hands and turned slightly to one side. Before beginning artificial respiration, the operator, as with the other methods, should see that the air passages are clear. The patient's tongue is made to fall forward by giving several light blows or slaps with the flat hand between the shoulder blades. The tongue need not be

The whole operation occupies 6 seconds which gives a frequency of operation of 10 times per minute. It should be noted that the pressure applied during the expiration movement should be just sufficient to lightly compress the chest. A pressure of 12 to 15 kgf will usually suffice for the average patient.

Inflation method: Expired air resuscitation
This method is also called the Biblical method. It is probably the earliest method of artificial respiration. It was only in recent years that it came to be recognised as an effective method of resuscitation of asphyxiated persons in mines. In this method, the rescuer blows air forcibly into the patient's mouth or nose to inflate his lungs and simulate the act of inspiration. Expiration will automatically occur due to the elastic recoil of the chest walls. Investigations have shown that an oxygen content of 16 per cent in the exhaled air of the first-aider is adequate for successful resuscitation and the carbon dioxide in the expired air helps to simulate the respiratory centre of the patient.

Before applying the mouth for resuscitation, the patient must be laid on his

the bottom is inserted into the mouth. The mouth is then sealed by pressing both the thumbs against the thumb pieces of the shield. The first-aider can then blow the expired air into the lungs of the patient through the tube (Fig. 7.31).

If lung ventilation is difficult due to blood, mucus or vomit, the original position of the patient should be changed, who should be turned to one side and placed in a stable lateral position. The mouth should then be cleaned by the forefinger or by using the Draeger Orosucker (Fig. 7.32) which consists of a plastic bottle of about 100 ml capacity with a mouthpiece and a catheter. By applying suction with the mouth (Fig. 7.33), a negative pressure up to 4 m water column can be created in the bottle and maintained by a built-in rubber valve.

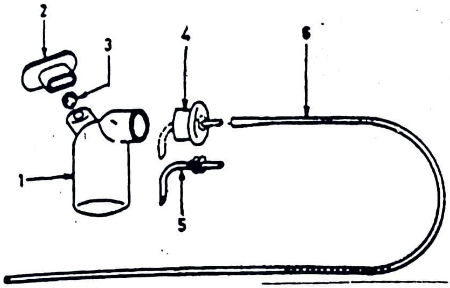


Figure 7.32 Draeger Orosucker.

Cardio-pulmonary resuscitation (CPR)

This is performed in cases of cardiac arrest in which the heart does not beat and pulse is not present. It combines expired air resuscitation and external cardiac compression. If during expired air resuscitation, a pulse is not present, external cardiac compressions must be started immediately.

The patient is laid on a hard flat surface and the first-aider then determines the pressure point for cardiac compressions by locating the bony tip of the breast bone (sternum) with his ring finger, and then placing two fingers just above that point. He places the heel of his hand adjacent to the fingers and then places the second hand on top of the first. Positioning his shoulders directly over the patient's breastbone, he presses downwards keeping arms straight so that the sternum is depressed 38 to 50 mm exerting about 40 to 50 kgf of pressure on the breast bone. The time spent in depressing and releasing the sternum must be the same.

If there is one first-aider, one-person CPR, compressions should be carried out at a rate of 60 to 90 per minute with two breaths with the nostrils pinched off (lung ventilation) after each 15 compressions in a row.

the lungs is expelled (expiration). For inspiration, the stomach and the chest (Biomotor) or the arms (Inhabad) are raised. In the second group are included apparatus which provide automatically-controlled forced-breathing resuscitation. An oxygen–air mixture is blown into the lungs during the inspiration phase and sucked out or expelled by recoil of the respiratory muscles and lungs during the expiration phase.

The forced breathing-type resuscitators are of two types: the pressure cycling type and the volume cycling type.

In the pressure cycling–type resuscitators, the apparatus forces oxygen through the mask into the lungs of the victim during the positive pressure cycle and pulls the air out by negative pressure. When oxygen is forced into the lungs under positive pressure, the pressure inside the chest cavity becomes greater than the pressure anywhere else in the body and, in consequence, the flow of blood into the chest from the peripheral veins is impeded thereby reducing the cardiac output sometimes to dangerous levels. A person can usually survive as much as 20 mm Hg continuous positive pressure in the lungs but exposure to pressure greater than 30 mm Hg for more than a few minutes may cause death. It is for this reason that most resuscitators are fitted with safety valves that prevent the positive pressure from rising above + 14 mm Hg and the negative

well- proven automatic equipment for resuscitation. It provides for automatic cycling of breathing by blowing an oxygen–air mixture into the lungs of the victim and removing the carbon dioxide-charged exhaled air out of the lungs by suction under pre-set positive and negative pressures. With simple adjustments, it can also be used for extraction of secretions from the respiratory tract and for oxygen inhalation.

Figure 7.35 shows the principal parts of the Pulmotor. It essentially consists of one or two 2.5-litre capacity steel oxygen cylinder(s) containing oxygen at a filling pressure of 200 bar, a pressure-reducing valve, a distribution manifold with three shut-off valves and a safety valve, a change-over valve for 50 or 100 per cent O_2, a cycling or switching valve unit, a face mask, an aspirating injector, a secretion bottle, an oxygen economizer breathing bag, an inhalation mask, and four sizes of oral airways. The reducing valve reduces the high-pressure oxygen to a constant operating pressure of 5 bar. Of the three shut-off valves, one is connected to the aspirating injector, another to the oxygen inhalation mask, while the third is connected to the switching unit. The switching unit is the soul of the apparatus. The changeover valve is normally set for 50 per cent O_2, the remaining the lungs is expelled (expiration). For inspiration, the stomach and the chest (Biomotor) or the arms (Inhabad) are raised. In the second group are included apparatus which provide automatically-controlled forced-breathing resuscitation. An oxygen–air mixture is blown into the lungs during the inspiration phase and sucked out or expelled by recoil of the respiratory muscles and lungs during the expiration phase.

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being atmospheric air sucked by an injector. It is set for 100 per cent O_2 in a toxic atmosphere. Provided that the respiratory tracts are clear and the mask fits tightly, the respiratory air flows into the lungs of the victim producing a clearly visible expansion of the thorax. As soon as the pressure of about + 20 mbar is reached in the lungs, the inspiration phase is completed and the switching unit switches over automatically to the expiration phase and begins to suck air out of the lungs. When the required negative pressure of –10 mbar is reached towards the end of the expiration phase, the appliance switches over to a new inspiration phase.

The switching rate depends on the size of the lungs and on the oxygen flow-rate. If the switching is too fast, the oxygen-supply must be reduced. The negative pressure at the end of expiration assists the blood circulation but in special cases a special attachment is supplied for cutting it out during the expiration phase.

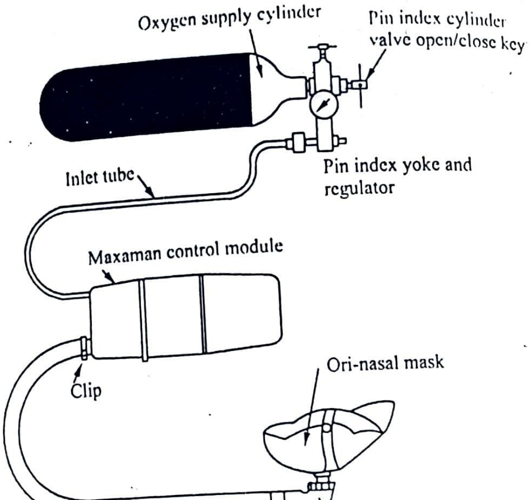
Before resuscitation, it is necessary that the patient is laid flat on his back with his head tilted backwards and turned to one side. After the pharyngeal cavity is cleaned, the lower air passages are cleaned by using the aspirating catheter of the Pulmotor aspirator. As soon as the air passages are free, an oral airway is inserted into the mouth to prevent the falling back of the tongue. The cylinder valve and the shut-off valve for resuscitation are then opened and the

to the victim's demands. It has been specially designed for use in irrespirable atmospheres wherein the victim may or may not be breathing when located or during the rescue operation. The rescuer need not know whether the victim is breathing or not. It is based on the mandatory minute volume lung ventilation technique.

The resuscitator consists of a compressed-oxygen supply cylinder with a pin index cylinder yoke, a first-stage pressure regulator and pressure relief valve, a control module connected to the pin index cylinder yoke by a small-bore tube and to a ori-nasal mask through a non-breathing valve, and a head harness (Fig. 7.36).

The first-stage pressure regulator reduces the oxygen pressure to a value between 4 and 7 bar before it flows to the second-stage pressure regulator in the control module.

The control module is an oxygen-powered automatic ventilator combined with a monitored demand facility. It consists essentially of three chambers. The central stainless steel chamber constitutes the reservoir of the system and contains tubing connecting together the logic units A and D and the demand valve. All the logic units and the second-stage pressure regulator are contained in a plastic covered chamber at one end of the stainless steel chamber while the demand valve, safety relief valve, and an obstruction whistle and indicator are contained in another plastic covered chamber at its other end. The inlet and outlet tubes are connected to the latter end.


Oxygen supply cylinder
Pin index cylinder valve open/close key
Inlet tube
Pin index yoke and regulator
Maxaman control module
Clip
Ori-nasal mask

In trials conducted by the erstwhile National Coal Board [88] on healthy subjects breathing normally, the best performance of Maxaman was obtained when the relief valve settings were 250 mm WG, 800/900 for the tidal volume, and 9/10 cycles per minute for the frequency.

The Maxaman set to operate only on demand has a potential application for escape purposes in an emergency.

7.4 Inhalators or Oxygen Reviving Apparatus

Inhalators or oxygen reviving apparatus supply oxygen to the patient in response to his own breathing demands. They are used in cases of asphyxia where the casualty has at least partial control of respiration but is not getting enough oxygen, or where the casualty has ceased to respire but has regained consciousness after successful resuscitation. It must be borne in mind that the quantity of oxygen available with a reviving apparatus is very small, usually 280 to 420 litre which is sufficient only for 20–30 min. duraton of use.

There are different types of inhalators available in the market. All are similar in construction as well as in operation but differ only in the method of oxygen supply and auxiliary attachment for reviving two or more casualtie of 50 per cent air and 50 per cent oxygen. A resuscitator with inhalation mask and breathing bag is an ideal appliance for oxygen inhalation.

7.4.2 Inhalators with lung-governed dosage

These are entirely of the lung-governed type available in different models with one, two, or four supply points or outlets. The apparatus with one supply outlet consists of an oxygen cylinder of 2.6-litre capacity charged to 150 bar pressure, a bypass valve, a lung-governed admission valve, a pressure gauge, a flexible breathing tube, and a moulded rubber facepiece with an expiratory valve. The lever(s) of the lung-governed valve is (are) enclosed in or protrudes (protrude) into a flexible bag protected in a protective casing. The lung-governed valve is opened when the bag is emptied by inhalation. The bypass valve enables the oxygen to be delivered as a continuous feed direct to the facepiece in cases of respiratory failure or faint respiration. A well-known inhalator still widely used in the Commonwealth countries is the Siebe Gorman Novox Model 1.

(2) To draw mutual-aid plans under which neighbouring mines and mines of neighbouring compances, after occurrence of a major mine disaster, assist each other on a reciprocal basis;

(3) To train directors or in-charges of rescue operations, leaders/captains, and apparatus mechanics or attendants in rescue and disaster techniques for mine rescue brigades, and also to conduct refresher training courses for them;

(4) To supervise the training of mine rescue brigades at mines and, under certain circumstances, even train mine rescue brigades;

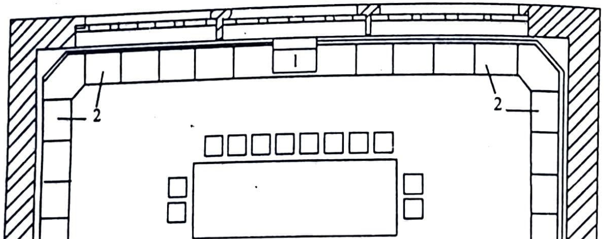
(5) To guide and supervise the conduct of rescue and recovery and reopening operations in mines;

(6) To maintain a permanent rescue corpse in constant readiness where the station is required to help in conducting rescue and recovery operations in mines falling within its radius of action;

(7) To store special types of equipment and apparatus for firefighting and mine rescue work;

rescue brigade. It comprises an apparatus room, a maintenance room, a cap lamp charging rack, modern hand-held electronic methanometers, CO monitors, and a test room. Figure 4.37 gives a layout of a mine rescue station or rescue room with a workshop for 20 self-containg breathing apparatus [89].

The central and district rescue stations should be located so that the radius of action does not exceed 25 to 35 km. All stations must efficiently cooperate with one another in emergencies.



The establishment, maintenance, and management of the rescue stations other than those at the mines are usually vested in a Rescue Station Committee appointed by the government, which includes representatives of the mines inspectorate, mine operators, mine employees and miner's union. The cost for establishment, equipping, maintenance and control of the rescue stations is met by levying on the mine operators an excise duty per tonne of coal or mineral mined.

The first rescue station is India was established at Oorgaum in the Kolar Goldfield in the year 1923 to serve four gold mines at Kolar. In the coal mining industry, the first rescue stations were established in the year 1941 at Dhansar in the Jharia Coalfield and at Sitarampur in the Ranigunj Coalfield. After nationalisation in 1971 and 1973, several rescue stations came into existence in mining areas in different geographic regions (45 rescue stations in the coal mining industry and 6 in the metal mining industry). The Mine Rescue Rules 1985 placed all the rescue stations under the respective companies in whose areas they existed for their administration, control, and operation for mine rescue coverage.

In some countries, mines may totally rely on state-sponsered rescue teams

7.5.1.2 Station apparatus and equipment

The mining regulations specify the minimum safety apparatus and equipment of the officially-approved type that should be kept and maintained at every rescue station. The equipment of a rescue station comprises the equipment stored at the station and a mine rescue van which is kept fully stocked and ready for use as a mobile rescue station whenever an emergency arises. A rescue station is equipped with self-contained oxygen breathing apparatus and necessary equipment for testing such apparatus, a portable supply of pressurised oxygen or oxygen-generating chemical as applicable to the supplied breathing apparatus, tins of absorbent chemicals, all-service gas masks, oxygen self-rescuers, gas detectors for CO and CH_4, oxygen indicators, safety lamps, (flame and electrical), bird testers with oxygen cylinders, permissible flashlights, electric cap lamp charging rack, resuscitators and inhalators, first-aid materials, stretchers, safety wear, portable mine rescue communication system or a sound-powered or any approved communication system, a mobile inert gas generator or a mobile nitrogen inertisation plant, an emergency power generator, a large-diameter hole drill rig, a rescue cage/capsule, a diesel–electric

decision-making, who will be responsible for the safety of the members. The members of the rescue brigades are drawn from both underground supervisory officials (deputies) and reliable workmen (miners and mechanics) who must have been employed underground for a minimum period of one year within the past five years. They are carefully selected on grounds of their coolness of temperament, powers of endurance, general suitability for work, age, and medical fitness as may be prescribed, and are made to undergo a course of instruction and training as prescribed in the regulations. The director and the deputy director are selected from among suitable technical supervisory officials who should have already worked for sometime as members of the mine rescue brigades. They should undergo a prescribed course of training at a central rescue station before they officially head the mine rescue organization. The director has responsible charge and control of the rescue organization and his appointment must be approved by the central rescue station and the mines inspectorate. In the UK, the director of operations may be a senior official on duty or a senior official of the area or central headquarters.

For summoning the rescue men employed underground, telephones have proved to be the simplest and quickest means of sending warning messages in emergencies in mines. Fire warning systems may also be used.

For summoning the rescue men on off-duty, telephone, sirens, messengers, amateur radio systems and portable radio-frequency phones may be used. Sirens and messengers as means of communication are seldom used today.

The notification of the on-duty and off-duty rescue men is done according to a pre-planned scheme in which the order or priority of notification and the method of communication are laid down.

7.5.2.2 Mine emergency notification plan

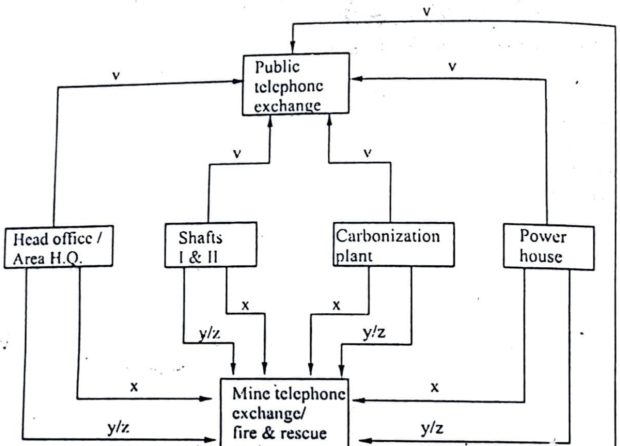
There are two types of messages for notifying officials and summoning help, 'Action Messages' and 'Warning Messages'.

'Action Messages' are addressed to persons whose attendance is required at once or who have to carry out duties immediately as outlined in their 'duty cards'. An action message chart must be prepared indicating the order in which

(19) Canteen Manager;
(20) Public Telephone Exchange;
(21) Colliery Engineer

Warning messages should be sent to the following:

(1) Central/District Rescue Station;
(2) Mines Inspectorate;
(3) Adjoining mines for assistance under mutual-aid plan;
(4) Head office of the mine/Area/Headquarters;
(5) Police Authorities;
(6) Doctors;
(7) Local hospitals;
(8) Connecting and adjoining mines endangered;
(9) Mine Safety Council/Commission;
(10) Mineworkers' Union; and
(11) Overmen and Deputies Union.
(12) District Magistrate



Rooms should also be provided for accommodation of the director/superintendent or the man in-charge of operations, rescue brigades from outside the mine, doctors, the injured, the dead, and for storage of food.

Experience with numerous rescue and recovery operations following mine disasters has shown that temporary underground and surface organisations should be formed as soon as mine rescue men comprising at least one brigade and other local mine officials and selected employees have assembled after their notification of an emergency at the rescue station or rescue room at the mine or surface rescue base, and at the mine office respectively.

If, after a mine rescue alarm underground, the rescue men present in the mine have assembled at a pre-planned central meeting station underground, their services as the first responders may be utilized, under certain circumstances, to start rescue and recovery operations or to do some exploration work in order to throw some light on the situation. In such a case, breathing apparatus must be transported from the surface to the meeting station by other rescue men and employees. Ordinarily, after the alarm, the rescue man first arriving at the surface rescue base performs the duties of the director or in-charge until the latter's arrival. On his arrival, the director must

When the task is too great for the temporary organisation to handle efficiently or when a devastating explosion occurs, the rescue and recovery operations by the temporary organisation should be continued until the arrival of additional assistance from the neighbouring mines, district and central rescue stations and other agencies when a permanent organization should be formed. The permanent organisation should take over the work started by the temporary organisation underground as well as on the surface, perform any other necessary functions, and complete the rescue and recovery work in the safest and speediest possible manner.

A running record of all important events as they occur should be kept for use as a Log.

It is extremely important for the mine management to properly handle the news media and the public during a mine disaster. All facts about the situation at specific points of time, covering important subjects, must be told as fast as they are available through spokespersons from the same platform.

7.5.3.1 Fresh-air base

7.5.3.2 First exploration ahead of fresh-air base

Explaration ahead of the fresh-air base is determined by the advisory committee after evaluation of conditions encountezed by rescue teams. As the fresh-air base is being set up, each brigade member, on orders from the captain, should carefully inspect and test his apparatus to ensure that the apparatus is air-tight and that its various parts are in safe working condition. After the apparatus is satisfactorily tested, each member reports to his captain who examines each member and his apparatus, opens the cylinder valve and connects the breathing tubes to the facepiece. The captain should have definite understanding with the director or the man in-charge of operations as to the extent of exploration, distance to travel, things to observe, and time to be taken in making the trip before he dons his apparatus aided and examined by one of the brigade members. The captain then makes entries of the names of the brigade members and the condition of their apparatus on a special card which he hands over to the director as he reports to the latter that he is ready to proceed. The director checks the entries, and, after comparing the time of his watch with that of the captain, gives the order for advance.

During exploration, the brigade members should examine the condition of roadways, mark with chalk their course of travel, location of bodies, and the farthest point of the trip. They should observe conditions and ascertain the extent and violence of the explosion. As an underground explosion drastically alters the prevailing ventilation patterns, new ventilation patterns must be established quickly so that effective rescue and recovery operations may be conducted. This can be done with brattice of impermeable fabric. In US mines, self-sealing brattice and parachute stoppings have been used in coal and non-coal mines with considerable success in rescue and recovery operations as well as in day-to-day ventilation changes during the normal working of the mine. The parachute stopping, like an ordinary parachute, is made of damage-resistant, light-weight fabric with regularly spaced straps around its perimeter that run to a common point. During erection, the common point of straps is attached to a roof bolt or some strongly fixed point and the stoppings is rolled out and lifted so that the ventilating air current inflates it into place sealing the perimeter. The self-sealing brattice is hemispherical and can be effectively used to replace damaged or destroyed stoppings. Such stoppings should be replaced as soon as

When operating in hot and humid atmospheres, the following precautions should be observed:

(1) Only such rescue persons should be selected as have good physical fitness and temperament for work in hot and humid atmospheres. Prior to undertaking rescue work, they should not have been engaged in hard physical work. Also, they should not have suffered from any organic disease or weakness in the recent past which would reduce their efficiency (for example, influenza, etc.) and their age should not exceed 45 years.
(2) The rescue men should wear minimum of light clothing.
(3) A brigade should not be allowed to operate for more than the safe period which should be decided on the basis of the condition of the atmosphere, travelling conditions in the roadways, and the tasks to be performed.

Maximum safe periods of operation under specified conditions of temperature and humidity have been recommended in some countries for the guidance of director of rescue operations.

Table 7.3 gives the maximum safe periods in the range wet bulb temperature 28° to 45°C as determined by tests in a climatic chamber in Germany wearin

Table 7.4 Maximum safe periods under difficult working conditions wearing ice-cooled jackets

t°_w \ Δt°C	0–2	3–4	5–6	7–8	9–10
			Minutes		
31	–	60	60	60	60
32	60	60	60	60	60
33	60	60	60	60	60
34	60	60	55	55	50
35	55	50	50	45	45
36	45	45	45	40	40
37	40	40	35	35	35
38	35	35	35	30	30
39	30	30	30	30	25
40	25	25	25	25	25
41	25	25	20	20	20
42	20	20	20	20	20

Table 7.5 give the maximum safe periods as recommended for use in the

unconscious victims are found, they should be carried out of the irrespirable atmosphere as soon as possible to the nearest point in fresh air or to the first-aid station at the fresh-air base. When transporting dead bodies, it should be remembered that the brigade should not ordinarily exhaust their strength in carrying the dead bodies any great distance in fresh air. 20 m relays is plenty for each pair to carry the loaded stretchers. Special precautions must be taken when handling dead bodies when they are much decomposed.

Table 7.5 Maximum safe periods under difficult working conditions for rescue personnel donning Proto compressed-oxygen breathing apparatus and wearing light clothing

In fully saturated atmosphere

t°_wC	Minutes
26.7	60
28.1	52
29.4	44
30.8	38

of the prevailing conditions and orders him to proceed in the same manner as the first explortion if further explorations are indicated before the fresh-air base is advanced. The brigade members hand over the apparatus to the apparatus mechanic, reporting any difficulties met with in their use. The progress of work done in each shift must be shown on a map of the mine showing area recovered, position of bodies, etc.

When it is advisable to snift the fresh-air base to a predetermined point, the reserve brigade proceeds to erect temporary stoppings, ventilate all open accessible workings by suitably conducting the ventilating current along the most direct paths known, and perform other necessary operations.

The above procedure of alternately exploring ahead of the fresh-air base and moving the fresh-air base should be continued until the entire area affected by the fires or explosion has been recovered.

7.5.3.4 Communication in rescue work

The director or man in-charge of operations at the first-air base must be in nication with the exploring brigades. The brigade members

7.5.5 Mine rescue through large-diameter drillholes

Large-diameter drillholes and bored shafts have been successfully used in Germany, Holland and the USA, for rescuing miners trapped alive underground after a rock/coal outburst or inundation. Special rotary and down-the-hole drills have been developed for the purpose. Rescue drilling may be carried out from suitable underground workings or from the surface depending on the location of the trapped miners. A probe/pilot hole of smaller diameter is first drilled to have access to the trapped men which is then reamed to a large diameter of 500 mm. It is possible today to drill rescue holes or shafts from the surface to a depth of 300 m through hard rock within a week. The trapped miners can be expeditiously and safely rescued through the drillhole by means of a rescue capsule or cage. For the first time in India, 65 miners trapped in Mahavir Colliery, Ranigunj, India, due to sudden irruption of water from water-logged workings in an overlying seam were rescued on November 15, 1989 by means of a rescue capsule through a 600-mm borehole.

Rescue of persons trapped underground after a mine disaster requires a plan and

7.5.6 Barricades

Barricading has proved to be a very useful technique for saving lives of trapped miners after a serious mine fire or explosion. A temporary shelter is constructed by the trapped miners out of available building materials in a nearby dead-end heading or location which is still free of toxic gases and an airtight shelter can be constructed to conserve an adequate amount of air so that they can survive for a few days while awaiting rescue.

The following important guidelines may be followed when barricading:

(1) The trapped miners should collect all tools, timber, boards, brattice, buckets, water, and any other materials which might be useful.
(2) Every effort should be made to erect the barricade as air-tight as practicable. Where compressed-air line is not available, the miners should not unnecessarily move in the enclosed area but should either lie down or sit on the floor to conserve oxygen. Occasionally, one person should slowly

The rescue chambers should be designed keeping in view the following objectives:

(1) withstand low-intensity explosions,
(2) be designed as far as possible with a minimum height of 1.8 m and a floor area and volume adequate for the number of workmen likely to use them,
(3) be constructed of incombustible and heat-resistant materials; with temporary rescue chambers the construction materials should have at least 1-hour fire-resistant rating,
(4) be located at easily accessible places in an area, where possible, free of combustible material in accordance with the maximum safe travelling distance which can be covered with or without the aid of a SCSR but inaccessible to air containing noxious smoke, fumes or gases,
(5) be equipped with a means for maintenance of a healthy respirable atmosphere,
(6) be equipped with a sufficient supply of potable water,
(7) be equipped with a means of communication with surface

In mines which do not have compressed air underground, a mine environmental control system Survivair-E has recently been developed by Naschem, a division of Denel (Pty) Ltd, Rep. of South Africa, which will provide an alternative method of ventilating reclaimable rescue chamber [92].

Survivair-E combines two integrated systems comprising the replenishment of oxygen consumed by means of six oxygen-generating-units and the removal of metabolically produced carbon dioxide by means of two separate carbon dioxide scrubbing units.

The oxygen-generating system releases oxygen through thermal decomposition of sodium chlorate chemical by an ignition device. Six generating units are used, each generating 2900 litres of oxygen of 99.5% purity within 45 min after activation. The oxygen generators comply with intrinsic safety standards set out by the South African Bureau of Standards and certified for use in fiery mines.

Another solution to carbon dioxide scrubbing was obtained by the development of personal respirator (DIOXSCRUB-P™) with a replaceable filter canister with 90% scrubbing efficiency and weighing 650 g also manufactured by Naschem (Fig. 7.40).

7.5.8 Accidents to mine rescue brigade members

The causes of accidents to rescue brigade members are:

1. Faulty mouthpiece breathing, facemask not properly sealed;
2. Faulty breathing equipment and accessories;
3. Incorrect handling of equipment and accessories;
4. Carelessness in performance of tasks assigned;
5. Overexertion by making long trips rather than short ones, etc.;
6. Undertaking work without specific orders; and
7. Not heeding difficult climatic conditions in the area to be recovered.

CHAPTER 8

Water Inundations

Ever since mining began, most of the mines throughout the world had always to contend with water problems which often assumed serious proportions.

One distinguishes between two types of water influx in mines: the normal

8.1 Causes of Inundations in Mines

8.1.1 Inundations by surface waters

Inundations by surface waters may occur after sudden and abnormally heavy rains when the surface mine outlets such as shafts, inclines, and adits may get flooded or subsidence fractures reaching the surface get submerged. They are very dangerous as large quantities of water break into the mine within a short time preventing escape of workmen. Surface drainage must be designed to cope with storm rainfall run-off conditions.

8.1.2 Inundations from overlying strata

Inundations from the overlying strata can occur under one or more of the following conditions: not allowing water to pass through & penetrate,

8.2.1 Surface measures

They can be further divided into:

(a) measures against flooding of mine main entries or outlets; and
(b) measures against seepage of surface waters.

The selection of one or more remedial measures is governed by the cost–benefit ratio and the method of financing.

8.2.1.1 Measures against flooding of mine main entries

These consist of the following:

(1) Locating shaft sites away from faults especially formed by major tensional stresses;
(2) Laying mouths of shafts, inclines, and adits above the high-water mark and lining them for the first 20 m with watertight lining.
(3) Filling up with debris and other sealing materials all abandoned shafts and boreholes which have ceased to serve any purpose with the existing mine;

anomalous structures such as fractures, joints or faults for the drilling and grouting programme.

(10) Leaving outcrop barrier pillars of adequate size;
(11) Selecting a method of extraction by which the strata subside uniformly without fracturing. For extraction under important waterways, hydraulic stowing is the best method of roof control; and
(12) Afforestation along river banks.

8.2.2 Underground measures

These protect life and property against inundations. They consist of the following measures:

1. General lowering of the water table to below the level of the workings by borehole pumps was extensively applied for a long time for opencast mines but it suffers from the disadvantage that a stage might be reached when further de-watering would severely interfere with water resources, wells,

(1) The degree of uncertainty regarding the exact location of the old workings;
(2) The thickness of the bed;
(3) The nature of the bed whether it is friable, contains dirt partings, or has been crushed and fractured by folding;
(4) The inclination of the bed and the direction of the pillars in relation to the line of full dip;
(5) The maximum hydrostatic head;
(6) The nature and condition of the adjacent strata whether pervious or broken;
(7) The method of extraction and roof control on the side of the pillar;
(8) The presence of faults, their throw and direction; and
(9) Effect of mining other beds.

There is, however, no general rule or formula for determining the size of safety pillars as no rule can possibly serve all cases.

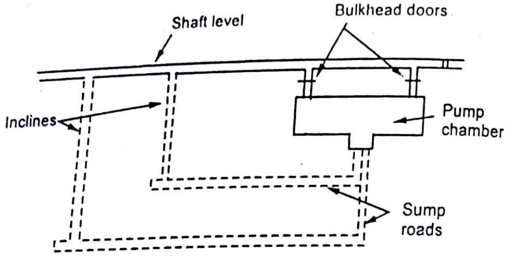
4. Supporting roadways not coming under the influence of mining operations

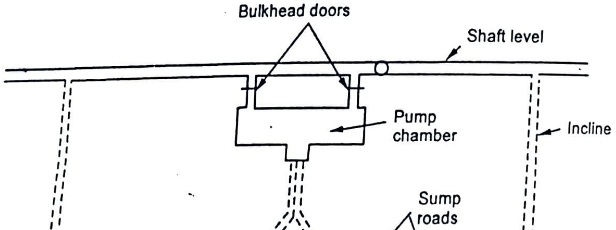
8.2.2.1 Mine sumps

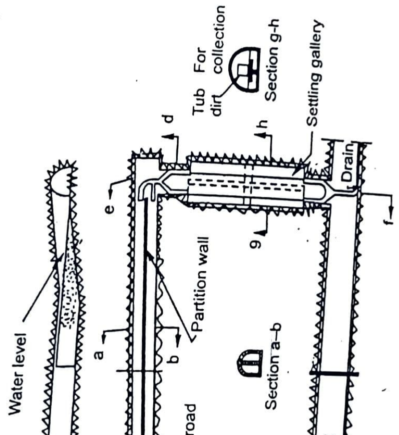
Every mine should have adequate sump and pumping capacities to deal not only with the normal water influx but also with any sudden inflows or irruptions of water or powerful feeders that may be encountered so that mine safety is assured. There should be sufficient power available to pump out all the water inflow to the surface.

Pumping systems vary from mine to mine depending on the rate of water influx, depth of mine, mine layout, and whether hydraulic winning and /or hydraulic stowing is practised. A mine may have a main pumping station at a deep level and water from upper levels may be allowed to gravitate to the main sump for pumping to the surface. Water may also be intercepted at various levels and pumped either direct to the surface from each level or between one or more levels to a main pumping station at the upper level. One characteristic feature of the main pumping station is that the main pumps and main sump are located near the pitbottom of a main surface shaft. In hydraulic-stowing mines, small pumping stations are established with settlers/settling tanks, which pump water to the main pumping station.

The capacity of a mine sump constructed near the pitbottom is provided by the sump roads of uniform cross-section which should be fundamentally driven below the main haulage level (about 4 m) in order to offer maximum


Shaft level
Bulkhead doors
Inclines
Pump chamber
Sump roads

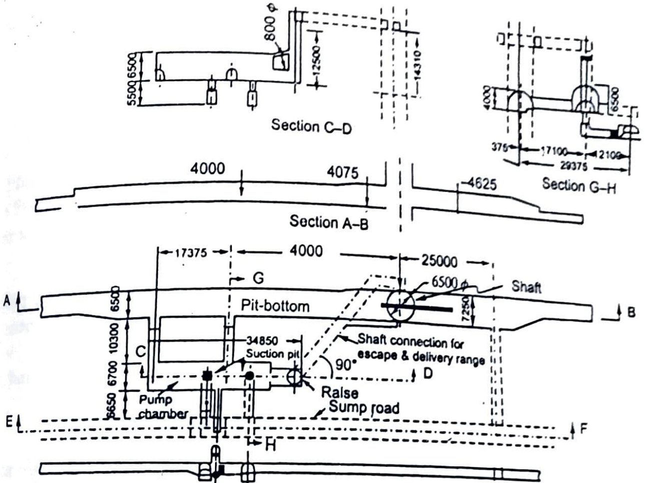

Bulkhead doors
Shaft level
Pump chamber
Incline
Sump roads


Tub For dirt collection
Section g-h
Settling gallery
d
h
Drain
e
f
g
Partition wall
Water level
a
b
Section a–b
road

As the drains which feed water to the sump roads carry a lot of solid particles and suspended dirt, a settling gallery or settling sump must be provided to settle out most of the sediments so that pump maintenance is minimised. Long shallow settling sumps are more commonly used than short deep ones. To facilitate sludge removal, they must be divided into two parts so that cleaning can go on in one while the other receives the inflow. If the pumps are well designed, comparatively little sludge will settle in suction pits in which the pump suctions are placed.

A convenient method of cleaning the sump is by means of a scraper which drags the dirt up an incline, or a track can be laid on the bottom of the sump or above it if the sump is deepened below all along the track and the sludge can be shovelled into a car by hand. The car is then hauled up an incline to the level.

Only electrically-driven conventional horizontal multi-stage turbine pumps having motors of high voltage, 3000 V, 50 cycles, must be used at the main sump in deep mines. Where two pumps are installed, each pump should be of such a capacity that it can deal with water influx over a period of 24 hours in 20 operating hours if pumping is to be done in all the three shifts. Where the sump capacity is large, large-capacity pumps may be used to operate on night shifts, week-ends, and holidays when tariff is low as an energy conservation measure.


800 ϕ
12500
14310
6500
5500
Section C–D
4000
6500
375
17100
2100
29375
Section G–H
4000
4075
4625
Section A–B
17375
4000
25000
G
6500 ϕ
Shaft
A
B
Pit-bottom
7250
6500
10300
6700
6650
C
34850
Suction pit
Shaft connection for escape & delivery range
90°
D
Raise
Sump road
Pump chamber
E
F
H

(3) When approaching waterlogged workings and draining off the water by means of boreholes;
(4) To limit the amount of pumping by allowing water in worked-out areas to accumulate behind the dams;
(5) To reduce or eliminate acid water being discharged in mines with the acid mine drainage problem;
(6) To seal off operating mines that will close in the future;
(7) To isolate acid-producing zones in a mine where they can be identified from active mine workings where natural barriers cannot be provided; and
(8) In most cases, to withstand the maximum hydrostatic pressure that may develop.

The problem of sealing off with dams can minimised or avoided by proper mine planning which would minimize water pressures so that less expensive dam constructions would be required.

When selecting a suitable site for a dam, the purpose of the dam and the hydrological conditions must be taken into consideration.

The following points must be borne in mind:

The selection of a type of dam is governed by the physical characteristics of the site and its preparation time, safety, availability of finances, and practical considerations.

Depending on the materials used in their construction, the dams may also be classified into clay, timber, brick, brick-and-concrete, and concrete dams. Brick-and-concrete, and concrete dams are most commonly constructed in mines because of their reliability and the small space required for dams of great strength.

8.2.2.2.2 Design of dams

The factors governing the design of a dam are:

(1) Size of roadway;
(2) Nature of the adjacent strata;
(3) Estimated water pressure;
(4) Materials used in construction; and
(5) Form of dam.

The size of the roadway in which a dam is to be constructed is a determining factor in dam design. For the same head of water, a dam built in a larger

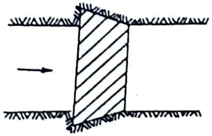


Figure 8.6 Brick dam.

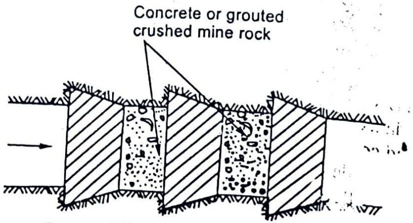
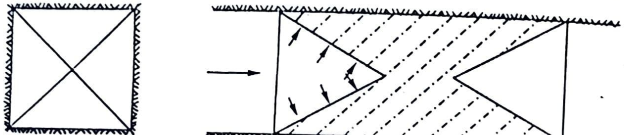


Figure 8.7 Brick dam for larger pressures.



The minimum length in the middle of the plug with hollowed out ends is given as t/4.

Concrete plug dams are successfully used in South African gold mines.

h dams (Figs. 8.9 and 8.10)

arch [illegible] distributes the thrust acting on it by arch action to the abutments in ne ground containing it. It may be designed as one or more arch rings each of which is considered to be subjected to a constant radial hydrostatic pressure.

The radial thickness of an arch ring may be approximately determined by using the cylinder formula and by considering the dam to be a segment of a thin cylinder subjected to an external water pressure.

$$t = \frac{pr_e}{\sigma_s} = \frac{pr_i}{\sigma_s - p} \text{(m)}$$

where p is the maximum water pressure (N/m²)

is the external radius of the arch ring (m)

When the safe compressive stress of the material used in construction is exceeded by the water pressure, a series of independent arch rings (usually two to four) will be necessary, each ring being designed for a water pressure of p/n, where n is the number of rings.

$$t = \frac{r_i}{\dfrac{n\sigma_s}{p} - 1}$$

the internal radius, r_i can be deduced from the relation:

$$r_i = \frac{b}{2\sin\dfrac{\alpha}{2}}$$

where b is the width of the roadway (m) and $\alpha/2$, the half central angle of the arch

= 20° to 30° in strong ground

= 15° to 20° in moderately strong ground

$$\sigma_s = \frac{\sigma_{ult}}{\nu}$$

where

σ_{28} is the 28-day compressive strength of concrete (N/m^2)

n is the time between completion of dam and application of full load on the dam (days)

σ_{ult} is the ultimate compressive strength of bricks (N/m^2)

ν is the factor of safety which may be taken as 1.5 to 2 for bricks and 2 to 3 for concrete.

The grade of the cement and strength of the bricks to be used must be properly selected. Aldis derived the following formula for determining the thickness of an arch dam:

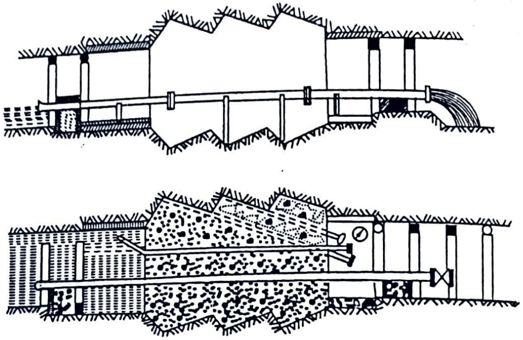


Fig. 8.11 Construction of flat concrete dam.

Steel anchors

the pipe is firmly embedded in the dam, it is provided with several flanges. A pipe 20 to 25 mm in diameter is also brought through the dam near the roof to vent air from the back of the dam as the water fills the roadway as well as to measure the water pressure. Owing to the difficulty of making the dam watertight at the abutments, pipes 30 to 50 mm in diameter, are also left in the dam for cement injections. Figure 8.11 gives the construction of a simple flat concrete dam [97].

Figure 8.12 gives the construction of a flat concrete dam suitable for water pressures of 12 to 15 bar (g) [98]. Steel anchors are inserted into the four sides of the roadway which are also injected with cement.

Figure 8.13 shows the construction of brick-and-concrete dam used in British mines. It consists of two 0.9-m thick brickwalls in cement mortar, about 3.6 m apart, with the space between filled with broken rock. Pipes of varying length are laid through the front wall as it is built up to inject cement into the loose rock to render it watertight.

8.2.2.2.4 Bulkhead doors or emergency dams

A bulkhead door is a watertight dam containing a strong steel door which is quickly closed in the event of a sudden

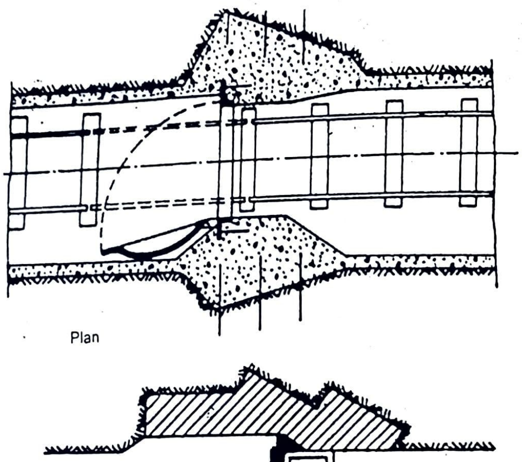


Figure 8.15 shows a bulkhead door in a parallel-sided concrete plug experimented in a deep gold mine of South Africa.

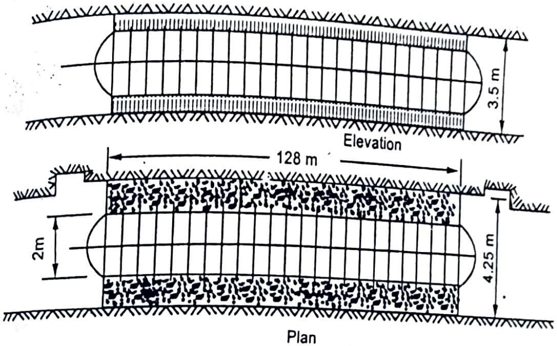

3.5 m
Elevation
128 m
2m
4.25 m
Plan

Figure 8.15 Bulkhead door in concrete plug.

Concrete cap

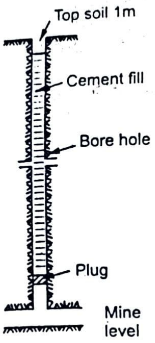


Figure 8.18 Hydraulic sealing of borehole.

8.2.2.2.5 Hydraulic sealing of shafts, adits, inclines, and boreholes
Figure 8.16, 8.17 and 8.18 show the methods of hydraulic sealing of vertical

8.3.1 Dewatering of waterlogged workings

The necessity of dewatering waterlogged workings arises:

(a) When a safety barrier between an active and an inactive mine is considered unsafe as a protection against a hydrostatic head that is more than a definite value;

(b) When it becomes necessary to drain water from an abandoned mine or mines into an adjacent active mine to operate with safety;

(c) When it is desired to extract reserves left in an abandoned mine filled with water from an adjacent active mine by draining water out of the abandoned mine into the active mine and utilizing the surface shafts of the abandoned mine for ventilation and manriding; and

(d) When an active mine or part thereof is inundated and the mine is to be quickly brought into production.

Besides ensuring complete safety, de-watering obviates the necessity of leaving barriers or providing dams for the protection of new mine workings

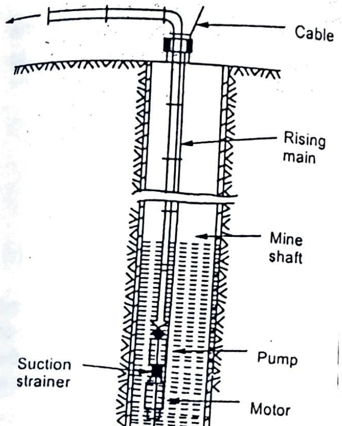
workings are directly fed from the surface and the inflow is considerable, dewatering may be inadvisable unless the country's economy demands it. This is because an expensive pumping plant may be required and the pumping cost over a long period may make mining unprofitable.

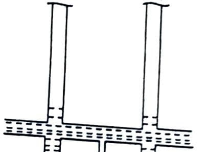
8.3.1.1 Methods of Dewatering

The method of dewatering to be adopted depends on whether an entire mine (active or inactive) is flooded, layout of the mine (in-the-seam or horizon mining), location and extant of waterlogged workings, time available for dewatering operation, available power supplies, and physical condition of the shaft(s) and mine.

Dewatering of a flooded mine in which water has risen into the shaft(s) may be done by suspended electrical vertical centrifugal/turbine pumps, air-lift pumps, or electrical submersible pumps installed in the shaft.

Where a standard vertical pump is used as dewatering pump, it is lowered as the water recedes, the discharge being connected to the pump column by a flexible rubber hose. In another approach, when the dewatering pump lowers





8.3.1.1.1 Dewatering by means of narrow mine roadways

This method is feasible only when it is possible to approach a waterlogged area sufficiently near to it and when the water pressure is less. The safe distance up to which the workings may be driven depends on the water pressure and the nature of ground. It can be determined by using the formula [94]:

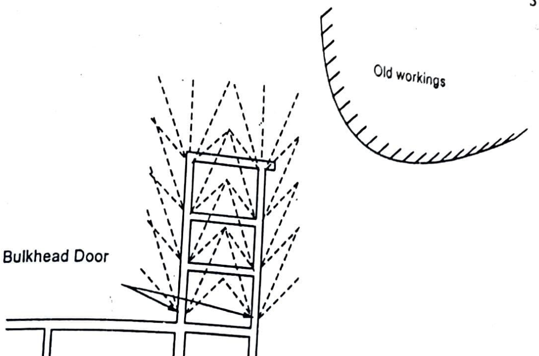
$$s = \frac{pb}{4\tau}$$

where

b is the length of the longest side of the roadway section (m)

p is the water pressure at the floor level of the approaching working (tf/m^2 or m WG)

s is the shortest distance between the approaching mine working and the water-logged area. It must be at least equal to the longest side of the roadway section when the direction of force is acting against the face, and equal to the roadway height when the direction of force is acting against the roadway side.



8.3.1.1.3 Precautions when approaching waterlogged workings

During drivage of dewatering mine workings and tapping of water, a numberof precautionary measures must be adopted. These include the following:

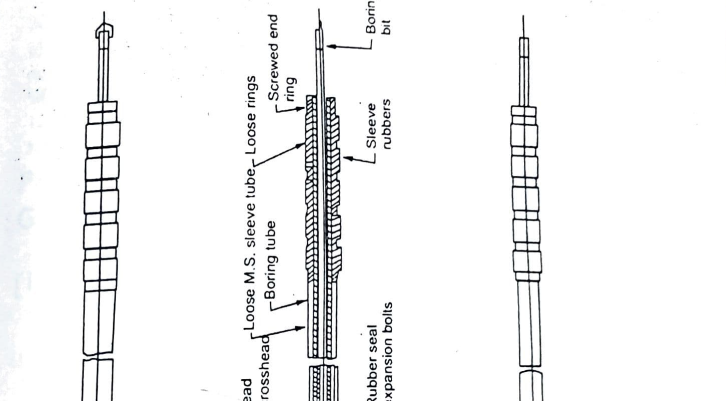
(1) For the safety of men engaged in dewatering operations, an escapeway, preferably a level one, must be provided. The escapeway must be properly supported, be free from obstructions, and be electrically lighted. If an escapeway is steeper than 3°, it must be provided with a water drain, a raised travelling way, and a rope railing.

(2) The dewatering mine workings must be well-ventilated to remove any noxious gases such as carbon dioxide and hydrogen sulphide that may occur. If through ventilation is not possible, provision for auxiliary ventilation must be made. Forcing ventilation system must be preferred.

(3) Open lights should not be used. Only electric safety lamps should be used.

(4) Only experienced miners should be employed who should be equipped with self-rescuers.

(5) For rescue of men overcome with gases, a rescue team equipped with respiratory protective apparatus should be stationed near the place of work. should be done by the rescue men



seal, expansion bolts, a debris cock, a cross-piece with rubber clamps, a fullway control cock, and a stuffing box and gland. The boring rods have a diameter of 38 mm (heavy type) or 33.3 mm (light type) and are made in the lengths of 1.52 m (5 ft) and 3 m (10 ft). The boring bits are tungsten-tipped wet drilling bits of 56 mm (2.19 inches), 73 mm (2.88 inches) and 95 mm (3.75 inches) diameter. The tungsten carbide-tipped reamers have diameters of 111 mm (4.38 inches), 121 mm (4.75 inches), 146 mm (5.75 inches), and 222 mm (8.75 inches). This machine has been in successful operation in the UK and India.

APPENDIX 1

Coalmines Regulations, 1957

Precautions Against Dangers from Fire, Dust, Gas and Water

[116] Classification of coal seams according to their degree of gassiness[1].

(1) After the coming into force of this regulation, all the coal seams shall be classified into different degrees of gassiness by the Chief Inspector or an Inspector assisted by such assistants and after such investigation as he

(4) Notwithstanding anything contained in sub-regulation (2) or (3), the Regional Inspector may, at any time, make investigations and re-classify gassy seam into the appropriate degree of gassiness.

(5) The Owner, Agent or Manager shall, at least once in every 30 days, examine the rate of emission of flammable gas in cubic metres per tonne of coal raised and the percentage of flammable gas in the general body of air as laid down in regulation 145. The results of every such examination shall be recorded in a bound-paged book kept for the purpose.

[116-A] General precautions against fire[1]

(1) No oil, grease, canvas or other flammable materials shall be stored in any mine in a fireproof receptacle. In case of workings below ground, greasy or oily waste shall be regularly removed to the surface.

(2) No person shall place or throw, or cause or permit to be placed or thrown any naked light or lamp on or near any timber, wooden structure, or other

except by the permission in writing of the Manager and only for a special purpose specified therein. All such permissions shall be recorded in a bound-paged book kept for the purpose: Provided that this sub-regulation shall not apply to boilers other than vertical boilers.

(6) A competent person shall, once at least in every seven days, inspect the top of all entrances to a mine, all opencast workings and any ground broken by extraction of coal in order to ascertain whether the precautions laid down under the regulation have been complied with, and for the presence of any fire that may have broken out or any indications thereof. A record of every such inspection shall be maintained in a bound-paged book kept for the purpose, and shall be signed and dated by the person making the inspection.

118 Underground precautions against fire

(1) (a) No timber or other combustible material shall be used in the construction of, or in connection with, any shaft-lining or room housing any machinery or apparatus belowground.

(5) All unused workings connected to the surface through a walkable entrance which is not permanently closed, shall once at least in every 30 days be inspected by a competent person for signs of illicit distillation of liquor. A report of every such inspection shall be recorded in a bound-paged book kept for the purpose, and shall be signed and dated by the person making the inspection.][1]

118-A Further precautions against spontaneous heating.[2]

The following further precautions shall be taken against the danger of spontaneous heating:

(1)

(a) The seam or section shall be worked in panels having independent ventilation in such a manner that it is possible to isolate one from another easily, if necessary. Where development has already b

(3)

(a) In order that spontaneous heating is detected in early stages, the air in the return airway of every de-pillaring district, and of every goaf which has not been isolated shall be:

(i) tested for percentage of carbon monoxide once at least in every seven days with an automatic detector of a type approved by the Chief Inspector; and

(ii) completely analysed once at least in every 30 days with a view to determining the ratio CO formed/O_2 absorbed.

The result of every such test shall be recorded in a bound-paged book kept for the purpose and shall be signed and dated by the person carrying out the test.

(b) If successive tests show any steady increase in the CO formed/O_2 absorbed ratio, suitable measures shall be taken to determine the site of the heating and to deal with it.

(c) Every de-pillaring district shall be inspected on every idle day, and all unused workings which have not been sealed off shall be inspected once at least in every seven days, by a competent dealing with or sealing off the fire or heating, shall be re-admitted in the mine until the fire or heating has been extinguished or effectively sealed off and an examination has been made by the Manager or by the Under Manager or Assistant Manager and the mine has been declared to be safe. A report of every such examination shall be recorded in a bound-paged book kept for the purpose and shall be signed and dated by the person making the examination: Provided that the Regional Inspector may, by an order in writing and subject to such conditions as he may specify therein, permit the employment in the mine of persons other than those required to deal with the fire or heating.

(b) The examination required by Clause (a) shall be made with an approved flame safety lamp and a cage containing suitable birds or other means of detecting carbon monoxide gas approved by the Chief Inspector. No additional light shall be used other than an approved safety lamp or torch.

(2) During the whole time that any work of dealing with or sealing off a fire or heating belowground is in progress:[1]

(a) a competent person shall be present on the spot throughout; and

(b) adequate precautions shall be taken to prevent danger to persons from flammable gases, flame, steam and ejected or rolling

carbonaceous material and exposed coal surfaces liable to heating, an adequate quantity of water at sufficient pressure for the purpose of efficient firefighting shall be provided and kept maintained;

(b) fire stations with a suitable supply of firefighting equipment shall be established and kept maintained at convenient points, both on surface and belowground; and

(c) sufficient supply of sand or incombustible dust or suitable portable fire-extinguishers shall be provided at:

(i) every entrance to a mine or district and at every landing and shaft bottom in use;

(ii) every place where timber, grease, oil or other flammable material is stored;

(iii) every engine room, electrical gear, driving unit of roadway conveyor, diesel engine, maintenance workshop and filling station, storage battery charging station; and

(iv) such other special places of fire risk as may be specified by the Manager.

(d) an up-to-date fire-fighting plan showing the water mains, taps firestations, pumping

[121 Apparatus for testing for carbon monoxide

In every fiery seam, there shall be kept at the mine, constantly available for uses two or more suitable birds or other means of detecting carbon monoxide gas approved by the Chief Inspector: Provided that the Regional Inspector may, by an order in writing require compliance with this regulation in the case of any other mine also.][1]

[122 Precautions when a fire exists.

(1) No person shall be employed in any seam:

(a) where a fire or spontaneous heating exists in a lower seam whether such fire has been sealed off by means of fire-stoppings or not; or

(b) where the seam has a common ventilation system with another seam on fire; or

(c) where the outlets or openings of the seam are within 60 metres of an active fire or spontaneous heating in a higher seam or on the surface in any ash heap or spoil heap or in any other heap or place or any other fire or spontaneous heating which cannot be controlled immediately; or

(d) where the outlets or openings of the seam are within 60 metres of such fire or spontaneous heating; or

(e) where the parting with an overlying seam on fire or in which spontaneous heating has taken place or with surface containing an

(4) (a) Every stopping erected to isolate or control a fire or spontaneous heating belowground[1] [or to seal off goaf or an area of old workings] shall be numbered, and shall be of adequate strength and so maintained as to prevent any leakage of air or gas through it. Where water is likely to accumulate behind any such stopping, there shall be provided in the stopping a suitable pipe or other device to drain away the water without permitting any leakage of air or gas etc.

(b) Where in any mine or part thereof the provisions of clause (a) have not been complied with or where, in the opinion of the Regional Inspector, the steps so taken are inadequate, he may give notice in writing to the Owner, Agent or Manager, requiring him to take such protective measures, within such time, as he may specify therein. In case of non-compliance with the requirements of the notice, the Regional Inspector may, by an order in writing, prohibit until the requirements of the notice have been complied with, the employment in the mine or part, of any person whose employment is not necessary for the purpose of complying with the requirements aforesaid.

(5) A competent person shall, once at least in every seven days, inspect all stoppings erected to isolate or control a fire or spontaneous heating belowground. During every such inspection, he shall ascertain the general condition of every stopping, check it

average concentration of airborne respirable dust in milligrams per cubic metre of air sampled by a gravimetric dust sampler of a type approved by the Chief Inspector by a general or special order exceeds 3 percent where working is being made wholly in a coal seam or where free respirable silica present is less than 5 per cent and the value arrived at by dividing the figure of fifteen with the percentage of free respirable silica present in other cases.

(3) (a) The Owner, Agent or Manager of every mine shall, within six months of the coming into force of this regulation and once at least every six months thereafter or whenever the Regional Inspector so requires by an order in writing, cause the air at every work place where respirable dust is evolved to be sampled and the concentration of respirable dust therein determined:

Provided that, if any measurement of any work place shows the concentration in excess of fifty per cent or seventy-five per cent of the allowable concentration specified in sub-regulation (2) (hereinafter referred to as "permissible limit") the subsequent measurements shall be carried on at intervals not exceeding three months or one month respectively.

Provided further that such measurements shall also be carried on

(f) All results of measurements of airborne respirable dust and all other relevant particulars shall be systematically recorded within fourteen days of the date of collection of samples, in a bound-paged book kept for the purpose, and every entry in the book aforesaid shall be countersigned and dated by the Manager within twenty four hours.

(4) When the dust-monitoring results have established that the permissible limit of dust concentration is being exceeded at any place, immediate steps shall be taken to minimise the emission of dust and to notify the Regional Inspector. If however, the average concentration of respirable dust in a series of 5 samples taken in 7 successive normal working shifts during the subsequent month exceeds one-and-a-half times the "permissible limits", the relevant operation or operations causing excessive dust shall cease. The operation or operations shall not be resumed and allowed to be carried on until improvements have been made in the prevention and suppression of dust and fresh sampling carried out immediately on resumption of the said operation or operations has established that such improvements have reduced the dust concentration below the "permissible limit":

Provided that, if the dust prevention and suppression device of any machinery or equipment fails to operate efficiently, the operation of the to the cutting edge to wet the cuttings or other equally efficient device approved by the Chief Inspector shall be provided and kept in operation to prevent the atmosphere being charged with dust. Where pneumatic drilling is performed, water shall be turned on before turning on compressed air to the drill. When however, drilling is done by hand, it shall be sufficient if holes are kept constantly moist during such drilling;

(c) every roadway on surface or belowground, where mobile mining machinery ply, shall be regularly wetted or shall be effectively treated with some other equally efficient agent to reduce dust being raised in the atmosphere to a minimum;

(d) no plant for the screening or sorting of coal and, as far as practicable, no heap of cinder, cement, sand, mortar or other dry and fine material, shall be placed within 80 metres of the top of downcast shaft or intake airway nor shall any such material be so handled as to make it airborne and drawn into any such shaft or such airway;

(e) in every mine belowground:

(i) no machinery or equipment which is likely to emit dust in excess of "permissible limit" shall be operated unless it is equipped with a suitable dust prevention and suppression device and unless such device is operating efficiently;

as to reduce the formation of dust to the minimum. Such material shall be also thoroughly wetted with water unless it is already wet or other effective means of dust suppression are used; and

(vii) unless, owing to special difficulties, exempted in writing by Regional Inspector in that behalf and subject to such conditions as he may specify therein, water in pipes, in sufficient quantity and under adequate pressure and independent of any pumping system, shall be provided and maintained so as to get maximum efficiency in the allaying of dust.

(f) no process of crushing, breaking, disintegrating, dressing, sorting, grinding, screening or sieving of coal, or any operation incidental thereto, shall be carried out at any mine unless efficient watering or other appropriate and effective dust control measures such as, but not limited to, isolation, enclosure, exhaust ventilation and dust collection are designed, provided, maintained and used.

(g) the exhausted air belowground, which contains dust in excess of the "permissible limit" shall be efficiently diluted and if necessary filtered so as to reduce the concentration of respirable dust therein below ten per cent of the "permissible limit" before being recircuited into workplaces.

(9) If any doubt arises to any matter referred to in this regulation, it shall be referred to the Chief Inspector for decision.]'

[123A Execution of measures for dust control[2]

(1) (a) There shall be maintained at every mine having workings belowground a *dust plan* on a scale having representative factor of not less than 1:2400. The dust plan shall clearly show (by distinctive colours, code letters and/ or numbers) the separate areas

- (i) which are naturally wet;
- (ii) which require treatment with water indicating the system of water pipelines laid down for the purpose; and
- (iii) which require treatment with incombustible dust at such intervals of 24 hours, 7 days, 14 days, 30 days or three months or other specified period as the case may be. (The intervals aforesaid shall be based on the results of analysis of routine mine dust samples collected from the areas concerned.)

(b) The areas aforesaid shall be clearly demarcated in the workings belowground by means of suitable notice boards or by other suitable means.

(c) No such incombustible dust shall continue to be used if it is found by tests which shall be carried out once at least every three months, not to comply with the foregoing requirements:

Provided that when the supply of incombustible dust used in a mine is not from a regular source, these tests shall be carried out whenever a fresh supply of incombustible dust is received.

(d) Where any place or part of the mine is to be treated with incombustible dust-

- (i) before treating with incombustible dust, all coal dust shall be cleaned, as far as practicable, from the roof, sides, floor, props, cogs, bars, brattice cloth or any other objects or structure or place on which coal dust may deposit, and all dust so collected shall be removed to the surface within 24 hours;
- (ii) incombustible dust shall be spread on the objects, structure and places aforesaid in adequate quantity and at such intervals as may be necessary to ensure compliance with the provisions of this sub-regulation;
- (iii) the cleaning of coal dust and spreading of incombustible dust shall be carried out in the direction of the flow of the air;
- (iv) a sufficient supply of incombustible dust shall be kept readily

(4) The Dust In charge shall also see-

- (a) that every part of the mine which, under these regulations, requires treatment with water, is thoroughly drenched or sprayed with water immediately before firing shots and also at intervals during the working hours so as to strictly comply with the provisions of regulation 123 (5) (b) (ii);
- (b) that every part of the mine which, under these regulations can be treated with incombustible dust, is so treated as to strictly comply with the provisions of regulation 123 (i) (b) (i); and
- (c) that the arrangements for treating with water or incombustible dust as aforesaid are maintained in good order][1]

[2][123B Check on Measures for Dust Control

(1) For the purpose of ensuring adequate treatment of coal dust as required under [regulation 123 and 123A][3], systematic samples of mine dust shall be collected, tested, and analyzed at intervals and in the manner specified in this regulation

(4)

- (a) Representative sample of dust shall be collected once in every 30 days from every zone, and for this purpose samples may be collected from different sections a, b or c in rotation such that during every such period of 30 days, all samples are collected from the section a or section b or section c.
- (b) Representative samples aforesaid shall be collected in a systematic manner irrespective of the cleaning and treating operations but shall in no case be collected within [20 hours]4[1] of cleaning and treating of any zone, section, or part thereof.
- (c) If the representative sample of mine dust from any particular zone shows that the provisions of [regulation 123A (IA) (a)][2] have not been complied with, immediate steps shall be taken to clean and treat whole of the zone so as to comply with the provision of [regulation 123A (A) (a)][3].

(5) In every travelling roadway and in every airway other than those specified in sub-regulation (2) (a), samples shall be taken in such systematic manner and at such intervals (not exceeding three months) that proper check is maintained on the efficiency of the treatment thereof in terms of [regulation 123 (1A) (c)][4].

(6)

institution approved for the purpose of regulation 16. He shall be designated as the "Sampling In charge" and shall be different from "Dust In charge" appointed under regulation 123A. No other duties shall be entrusted to this person except with the previous permission of the Regional Inspector.

(8) Within seven days of taking of each sample, it shall be sent for analysis and the result of such analysis, immediately on its receipt, shall be recorded in a bound-paged book kept for the purpose. Every entry in the book aforesaid shall be signed and dated by the Sampling In charge and be countersigned and dated by the Manager.

Explanation: A place in a mine is considered "Naturally wet throughout" if it is moist enough to keep the coal dust present at any time, on the roof, sides and floor and other objects at that place so that it is always combined with not less than 30% by weight of water in intimate mixture.]"

123 C. Stone-dust Barriers[1]

125 Recovery and exploratory work

(1) After an explosion of flammable gas or coal dust has occurred in a mine, only such persons as are authorised by the Manager or by the principal official present at surface, shall be allowed to enter the mine.

(2) When it is intended or proposed to re-open a mine or part thereof, which has been isolated, sealed off or flooded with water to deal with a fire or spontaneous heating, the Owner, Agent or Manager shall, not less than 14 days before the commencement of such work, give notice in writing of such intention or proposal to the Regional Inspector and the Chief Inspector.

(3) Where it is intended to carry on any exploratory work in a mine or part belowground likely to contain irrespirable atmosphere-

(a) party of less than three persons shall be allowed to proceed to carry out such work; and

(b) every such party shall carry a cage containing suitable birds or other means of detecting carbon monoxide gas approved by the bank of a river or canal or from the boundary of a lake, tank or other surface reservoir, except with the permission in writing of the Chief Inspector and subject to such conditions as he may specify therein.

(3) Every application for permission under sub-regulation (2) shall be accompanied by two copies of a plan and section showing the existing position of the working of the mine, the proposed layout of workings, the depth of the workings from the surface, the position and depth of any goaves in every seam in the neighbourhood, all faults, dykes and other geological disturbances and such other particulars as may affect the safety of the mine or of the persons employed therein.

Explanation-Where sand or alluvium are lying in the course of a river, canal, lake, tank or reservoir, the depth from the surface shall be reckoned from the surface of the hard ground underlying such sand or alluvium.

[(4) All workings made under sub-regulation (2) shall be clearly demarcated belowground.][1]

(5)

(b) any water-bearing strata, all useful information including the position, extent and depth of the abovementioned features shall be acquired and kept recorded and a scheme of working designed to prevent irruption of water or other liquid matter or any material which flows when wet shall be prepared and put into operation.

(3) Without prejudice to the requirements of sub-regulation (1) and subregulation (2), no working which has approached within a distance of 60 metres of any other working (not being the working which has been physically examined and found to be free from accumulation of water or other liquid matter or any material that is likely to flow when wet) and whether in the same mine or in an adjoining mine, shall be extended further except with the prior permission in writing of the Regional Inspector and subject to such conditions as he may specify therein.

Explanation-For the purpose of this sub-regulation, the distance between the said workings shall mean the shortest distance between the workings of the same seam or between any two seams or sections, as the case may be, measured in any direction whether

(6)

(a) The height of any working referred to in sub-regulation (3) or subregulation (5) shall not exceed 2.4 metres and there shall be maintained at least one borehole near the centre of the working face and sufficient flank boreholes on each side where necessary, boreholes above and below the workings at intervals of not more than five metres. All such boreholes' [drilled sufficiently close to each other to ensure that the advancing face will not accidentally hole through into a working containing water or liquid matter or any material that is likely to flow when wet and shall be] maintained and shall be constantly maintained at sufficient distance in advance of the working and such distance shall in no case be less than three metres. These precautions shall be carried out under the direct supervision of an official, having Manager's or Overman's Certificate specially authorised for the purpose.

(b) A record showing the exact height and width of such working, the number of boreholes driven, the length of each borehole, the places at which and the direction in which each borehole was driven, shall be maintained by the official referred to in clause (a) in a bound-paged

128 Intentional flooding

(1) When the Owner, Agent or Manager intends or proposes, by introducing water from the surface or from any other part of the mine, or from an adjacent mine, to flood any part of the workings of his mine, he shall give, in writing, not less than 14 days' notice of such intention or proposal, to the Chief Inspector and the Regional Inspector and to the management of all adjoining mines and of such other mines as might be affected by such flooding: Provided that the Regional Inspector may, by an order in writing and subject to such conditions as he may specify therein, permit such operation to be commenced on any day prior to the expiry of the said 14 days:

Provided further that the Regional Inspector may, by an order in writing, either prohibit any such operation or require that such operation shall not be commenced until such precautions, as he may specify, have been taken to his satisfaction.

Explanation-For the purposes of this sub-regulation, a mine shall have the meaning assigned to it under regulation 30.

(2) If the operations in respect of which notice is given under sub-regulation (1) are not commenced within 60 days from the expiry of the said 14 days, the notice shall be deemed to have lapsed and the provisions of sub-regulation

APPENDIX 2

Mines Rescue Rules, 1985

Chapter I Preliminary

1. **Short title, application and commencement.**

(1) These Rules may be called the Mines Rescue Rules, 1985.

(2) They shall apply to below/ground mines.

(3) They shall come into force on the 2nd April, 1985.

2. **Definitions.** In these rules, unless there is anything repugnant to the subject or context,

(a) "*Act*" means the Mines Act, 1952;

liquidator or receiver, but does not include a person who merely receives a royalty, rent or fine from the mine, or is merely the proprietor of the mine, subject to any lease, grant or licence for working thereof, or is merely the owner of the soil and not interested in the minerals of the mine; but any contractor or sub-lessee for the working of a mine or part thereof shall be subject to the Act in like manner as if he were an owner, but not so as to exempt the owner from any liability;

(m) "*Principal Official*" means the senior-most mine official in mining discipline on duty in the mine;

(n) "*Qualified Medical Practitioner*" means a medical practitioner who possesses any recognised medical qualification as defined in clause (h) of Sec. 2 of the Indian Medical Council Act, 1956 and who is enrolled on a State medical register as defined in clause (k) of that section;

(o) "*Rescue Room*" means a rescue room as established and maintained under Rules;

(p) "*Rescue Station*" means a rescue station established and maintained under Rule 3;

(q) "*Rescue-trained person*" means a person certified by the Superintendent to be rescue-trained person under sub-rule (1) of Rule 21];

and the Inspector in-charge of the region where the mine is situated (hereinafter referred to as Regional Inspector) in Form 1.

4. **Functions of Rescue Stations.** The functions of rescue stations shall include:

(i) imparting initial training in rescue and recovery work;
(ii) imparting refresher training to rescue-trained persons;
(iii) providing support by its own rescue teams and equipment in case of major accidents or long-lasting rescue and recovery operations; and
(iv) carrying out the functions of rescue room in respect of these belowground mines where there is no rescue room.

5. **Establishment and Location of Rescue Rooms**

(1) At every below-ground mine where more than 100 persons are ordinarily employed below -round and there is no rescue station within a radius of 35 km, the Owner shall establish and maintain on surface close to the mine entrance a rescue room:

Provided that it may be sufficient to provide one rescue room for a

7. **Qualifications, Experience, etc., of Superintendent**

After coming into force of these Rules, no person shall be appointed as Superintendent unless he

(i) is a rescue-trained person;
(ii) has 5 years practical experience of below ground work in mines; and
(iii) holds a First Class Manager's Certificate of a type not restricted to opencast workings.

8. **Qualifications, Experience, etc., of Instructors** After the coming into force of these rules, no person shall be appointed as Instructor unless he

(i) is a rescue-trained person;
(ii) has 3 years practical experience of below-ground work in mines; and
(iii) holds a Manager's Certificate of type not restricted to opencast workings.

9. **Selection of rescue-trained Persons for posting at Rescue Stations**

(1) After coming into force of these rules, the rescue-trained persons for posting at a rescue station shall be selected from amongst the rescue

(6) All breathing apparatus and every flowmeter shall be adjusted and tested and the purity of oxygen for use in breathing apparatus shall be tested in such manner as are laid down in Sch. IV.

(7) In case of an accident in a below-ground mine arising out of the use of any rescue apparatus, a written report thereof shall be sent, within 24 hours of such accident, by the Manager of the mine to the Chief Inspector and Regional Inspector in Form II appended to the Mines Rules.

(8) In case of an accident in a rescue station or rescue room arising out of the use of any rescue apparatus, a written report thereof shall be sent, within 24 hours of such accident, by the Superintendent to the Chief Inspector and Regional Inspector in Form II aforesaid.

(9) No absorbent charges, chemicals, self-rescuers and gas detector tubes shall be kept at a rescue station, rescue room, or below-ground mine for use beyond the expiry date, which shall be legibly marked on the boxes thereof.

Chapter 3 Duties and Responsibilities of Superintendent, etc.

(8) The Superintendent shall maintain in a bound-paged book, a diary and shall record therein the results of each of his inspections and also the action taken by him to rectify the defects observed during Inspection.

(9) The Superintendent shall submit to the Chief Inspector and to the Regional Inspector a detailed report on the functioning of the rescue station and rescue rooms under his control in the preceding year on or before the 20th day o February of the succeeding year with particular reference to
   (a) type and number of equipment and apparatus provided and maintained;
   (b) Superintendent, instructors, rescue room in-charge and rescue brigades in position as on the 31st December;
   (c) number of rescue-trained persons at each below-ground mine served by the rescue station and number of such persons given refresher practices and medically examined, with results thereof;
   (d) particulars of emergencies attended;
   (e) any other relevant matter.

13. Duties of Instructors

(i) Under the direction of Superintendent, the instructor shall

(a) impart course of instructions and practices to persons

(b) attend to messages, telephone calls, wireless and maintain a record thereof;

(c) maintain the rescue station in neat and tidy condition;

(d) maintain the equipment and apparatus kept at the rescue station in perfect order;

(e) perform rescue and recovery work in mines; and

(f) not leave rescue station without obtaining permission from the Superintendent.

16. Duties of Rescue Room Attendant

Every rescue room attendant shall

(a) not leave the rescue room until relieved by a substitute;

(b) not allow any unauthorised person to enter in or to take out any apparatus or equipment from the rescue room;

(c) attend to telephone calls and maintain a record thereof;

(d) maintain the rescue room in neat and tidy condition; and

(2) The Manager of a below-ground mine, where more than 500 persons are ordinarily employed below ground, shall also ensure that persons on a scale of one man for every 100 persons or part thereof are rescue-trained persons.

(3) It shall be the responsibility of the Owner to provide suitable accommodation

- (i) close to the rescue station to the Superintendent, Instructor, and members of rescue brigade;
- (ii) close to rescue room, to rescue room in-charge, rescue-trained persons, and attendants attached to it; and
- (iii) close to below ground mine entrance, to rescue-trained persons other than those specified in (i) and (ii).

(4) There shall be provided at a below-ground mine effective bell or other arrangements, as may be approved by the Regional Inspector, for immediate summoning of rescue-trained persons.

22. Medical Examination, etc. of rescue-trained Persons

Every rescue-trained person shall be re-examined once at least every twelve months by a qualified medical practitioner as may be designated by the Manager in accordance with Sch. VI, and, in case he is declared medically unfit, he shall cease to be a rescue-trained person with effect from the date he has been so declared.

23. Suspension of rescue-trained Persons

If, in the opinion of the Owner, Agent, Manager or an Inspector, a rescue-trained person is incompetent or is guilty of negligence or misconduct in the performance of his duties, the Owner, Agent, Manager or Inspector, as the case may be, after giving such person an opportunity to give a written explanation, suspend him from or debar him for undertaking any rescue and recovery work.

Chapter 5 Conduct of Rescue Work

25. Accommodation at the below-ground Mine for Persons engaged in Rescue Work

Whenever rescue-trained persons are engaged in rescue or recovery work at a below-ground mine, there shall be provided at such mine suitable accommodation for storage and charging of apparatus required for that work. Such accommodation shall be situated at convenient place near the entrance in use, and shall be properly ventilated and lighted. Adequate quantity of cool and wholesome drinking water shall be provided at such accommodation, and proper canteen facilities shall also be provided for the rescue workers.

26. Entry into below-ground Mines for Rescue or Recovery Work

(1) No person shall be allowed to enter a below-ground mine or part thereof which is unsafe for the purpose of engaging in rescue or recovery work, unless authorised by the Manager or in his absence by the principal official of the mine present at the surface. Only rescue-trained persons shall be

29. **Instructions to Leader**
Prior to sending a rescue team underground, the Superintendent or a person authorised by him shall give clear instructions to the Leader of the team as to where it shall go and what it shall attempt.

30. **Test of Apparatus**
Before proceeding below ground, the Leader shall test or witness the testing of self-contained breathing apparatus of the team for leakage. No such apparatus shall be used unless it is found safe. He shall check the equipment of his party and, immediately, before entering irrespirable atmosphere, shall make sure that all breathing apparatus are working properly.

31. **Duties of Leader below ground**
   (1) The Leader shall not engage in manual work. He shall give his attention solely to directing the team and to maintaining its safety. He shall examine the roof and supports during the journey, and, if there is any likelihood of fall at any place along the roadways, shall not proceed further until the team has made the place secure.
   the team together and shall not allow any member

keep the place given to him when numbering off. If the pace is too quick or if distress is felt, the member shall at one call attention to the fact.

   (5) Every member of a rescue team engaged in work with breathing apparatus in a mine shall obey the orders of the Leader of the team.

33. **Restriction of second Spell of Work**
No person shall commence a second or subsequent spell of work in irrespirable atmosphere without being examined and found fit by a qualified medical practitioner.

Chapter 6 Miscellaneous

34. **Obligation of Owner, Agent and Manager in a certain Situation**
Whenever emergency arises at a below-ground mine, whether served by a rescue room or rescue station or not, the Owner, Agent or Manager may seek assistance or additional assistance, as the case may be, from the nearest rescue room or rescue station and in such an event:
   (a) all possible assistance shall be promptly rendered by the rescue room

38. Repeal and Saving

(1) The Coal Mines Rescue Rules, 1959, are hereby repealed.

(2) Notwithstanding such repeal, anything done or any action taken under the said Rules shall be deemed to have been done or taken under the corresponding provisions of these Rules.

Schedule I

[See Rule II (1)]

Equipment to be kept at Rescue Room

		Nos.
A.	**Breathing Apparatus**	
	1. Two-hour self-contained breathing apparatus	54
	2. Short-duration self-contained breathing apparatus	6
	3. Absorbent charges	2,000
B.	**Resuscitating Apparatus**	
	1. Resuscitating apparatus	12
	2. Spare cylinders	8

G.	**Air- and Temperature-measuring Instruments**	
	1. Vane anemometer	1
	2. Velometer	1
	3. Clinical thermometer	2
	4. Whirling hygrometer or psychrometer	2
H.	**Telephone**	
	1. Genophone-Portable Telephone sets	2
I.	**Safety-and First-Aid Equipment**	
	1. Safety helmets	60
	2. Safety belts	6
	3. First-aid boxes	10
	4. Blankets	15
	5. Folding stretchers	12
	6. Self-rescuers	12
J.	**Fire-fighting Equipment**	
	1. CO_2 fire-extinguishers	4
	2. Foam-type fire-extinguishers	10
	3. Foam charges	50

B. Resuscitating Apparatus	
1. Resuscitating apparatus	3
2. Spare cylinders	5
C. Tube Apparatus	
(a) Pressure-type with bellows	
(b) Spare helmets	
(c) Pressure-type with fan	
D. Ancillary Equipment	
1. Oxygen cylinders	3
2. Oxygen pump (hand-driven)	2
3. Bobin meter	2
4. Universal tester (where necessary)	1
5. Pressure gauge testing device	1
6. Apparatus testing tool kit	2
7. Oxygen testing apparatus	1

E. Lamps, Etc., and Gas-testing Device

K. Miscellaneous	
1. Stop watch	1
2. Rain gauge	1
3. Lifeline	1 roll
4. Guide line	1000 m
5. (a) Evacuated sampling tubes	5
(b) Two-litre capacity stoppered sampling bottles	2
6. Siren (hand-operated)	1
7. Siren (power-operated)	1
8. Barometer	1*
9. Rescue van	

Schedule III

[See Rule 11(3)]

Minimum Equipment to be kept at the Surface of Mine where no

(iv) The pressure in the oxygen cylinder shall be measured.

(v) The rate of delivery of oxygen shall be measured by a flowmeter, bobinmeter, universal tester. If required, the reducing valve shall be adjusted to supply oxygen at a rate of flow specified by the manufacturers. A record of such measurements and adjustments shall be kept in a bound-paged register.

(vi) If an apparatus has been provided with an automatic demand valve, the pressure at which it operates shall be measured and, if required, adjusted, so as to operate within the limits recommended by the manufacturer.

(vii) Other tests and adjustments, if any, as recommended by the manufacturers shall be performed and a record thereof kept in a bound-paged register.

3. The purity of oxygen supplied for use in breathing apparatus shall be tested. No oxygen which is found to contain more than one per cent of impurities' shall be used.

(c) There shall not be any organic disease of the eyes, which is likely to affect distant vision within a period of one year.

(d) The person shall not be suffering from nystagmus or colour blindness.

(4) Hearing shall be good. Person shall be able to hear conversational voice from a distance of 3 metres.

(5) (a) Respiratory system shall be sound and free from any chronic laryngeal, bronchial and pulmonary disease.

(b) An X-ray examination shall be made once in two years and there shall not be any evidence of any pulmonary disease.

(c) The difference of chest measurements taken after full inspiration and full expiration shall not be less than 5 cm.

(6) (a) There shall be no evidence of cardiac or vascular disease.

(b) Normal heart rate shall not be above 90 or less than 60 beats per

Schedule VII

(See Rule 21)

Courses of Instructions and Practices

Part I-Initial Course

The course of instructions and practices shall be as follows:-

A. **Instructions:** Instructions in the following subjects-

(i) the general methods of dealing with fires below ground and the recovery of mines after fires and explosions;

(ii) the construction, use, repair, maintenance, and testing of the type or types of breathing apparatus provided, and the smoke helmets or other apparatus serving the same purpose;

(vii) the rapid establishment of communication.

Part II Practices and instructions after becoming competent

A. **Instructions**

Revision of all subjects included in Part 1.

B. **Practices**

At least eight practices with breathing apparatus in every calendar year of which at least four months shall take place in mines and the remainder in hot and irrespirable atmosphere in the Training Gallery at rescue station. As far as practicable, these practices shall be evenly distributed, so, however, that the gap between two consecutive practices shall not exceed four months:

Provided that, if a rescue-trained person fails to undergo a refresher practice over a period exceeding four months, the Chairman, Board of Mining

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